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THE SCHOOL OF MINES QUARTERLY

INDEX VOL. XXXIII.

November, 1911, to July, 1912.

All Abstracts are printed in Italics, and appear only under the headings Analytical Chemistry, Metallography, and Mining Decisions.

ABRAMS, HERBERT T. Pumping by Compressed Air 293
ALLEN, H. B. and CAMPBELL, W. Heat Treatment of a Nickel Steel..... 72
Alloys, Copper, Volatilization of Gold and Silver from. By E. F. Kern and A. A. Heimrod 34
ALUMNI AND UNIVERSITY NEWS 97, 193
Camp Columbia 97
Chemistry 103
Civil Engineering 194
Electrical Engineering 196
Enginering Drafting 197
Geology 99, 197
Mechanical Engineering 198
Metallurgy 199
Mineralogy 201
Mining 100, 202
Personal Items 103, 207, 315, 416
AMEND, C. G. and SHERMAN, H. C. Relation of Chemical Composition to Calorific Power in Wood, Etc. 30
ANALYTICAL CHEMISTRY, ABSTRACTS. By Elwyn Waller 172, 392
Alkalimetric Solutions, Standardizing 173, 392
Aluminum, Commercial 174
Ammonium Citrate for Phospahate Analyses 173
Anthracite Coal—Calorimetry 188
Antimony in Alloys—Rapid Volumetric Method 174
Arsenic and Antimony in Commercial Copper 175
Beryllium, Separation from Aluminum 181
Bismuth in Ores 177
Boric Acid in Preservatives 191
Bromine in Potassium Chlorate 184

Bromine—Qualitative 184
Bumping of Solutions 172
Cadmium in Spelter 177
Calcium Cyanamide—Lime-Nitrogen 187
Carbon in Steel 178
Carbon Disulphide in Oils 401
Cement, Ferric Oxide in 400
Cement—Rapid Analysis 190
Cerium—Separation by Permanganate 181
Chlorides, Chlorates and Perchlorates in Mixtures 184
Chloride in Commercial Bromides 400
Chloride in Presence of Bromide or Iodide—Qualitative..... 173
Coals, Volatile Matter in 400
Copper by Iodide Method 396
Copper—Methods of the Anaconda Copper Mining Co. 177
Copper, Nickel and Cobalt 398
Copper in Pyrites and Slag 177
Cupels—Loss of Silver 394
Cyanide Extraction of Gold—Control Sampling 395
Detonators and Fulminate Compositions 401
Electro-Analysis—Stirring the Solution 392
Ether in Metal Analysis 172
Ether—Test for Water 172
Ferric Oxide in Presence of Alumina 393
Ferroboron 400
Ferrocyanide 188
Ferrocyanide Determination 393
Ferrous Iron in Silicates 393
Ferrous Salts—Qualitative 393
Fluorides 187
Fluorides, Silicates and Fluosilicates—Qualitative 393
Formic Acid, Delicate Test for 188
Galvanized Iron 179
Gold Assay—Parting 394
Gold in Alloys 176
Graphite in Lubricants 189
Hyperol 392
Iron and Alumina Separation 174
Iron from Manganese—Separation 399
Iron from Vanadium—Separation 399
Iron Precipitations 399
Iron Titration 174
Lead Arsenate for Plant Spraying 188
Lead by the Chromate Method 176
Lead—Electrolytic Method 396
Lead, Higher Oxides of 176
Lead and Manganese—Delicate Test 392
Lead in Ores 176
Lime, Small Amounts of in Presence of much Magnesia 173
Lubricating Oils 189

Manganese—Bismuthate Method 397
Mercury—Volumetric 396
Molybdenum—Volumetric 184
Naphthalene in Gas 189
Nickel or Cobalt 394
Nickel Dimethylglyoxime 394, 397
Nickel in Ferrovanadium 400
Nickel in Steel—Comparison of Methods 179
Nickel Separation 397
Nickel—Separation from Zinc in German Silver, etc. 178
Nitrates and Nitrites in Water 192
Nitric and Nitrous Acids in Water 192
Nitrogen in Nitrates 188
Oils for Automobiles and Gas Engines 190
Oils, Mineral, in Fatty Oils 400
Palladium 182
Peroxidized compounds 188, 393
Phosphates—Colorimetric Method 186
Phosphoric Acid—Volumetric Method 186
Platinum—Gold—Silver Bullion, Parting 176
Platinum—Silver Buttons, Parting 394
Potassium by Cobaltinitrite Method 191
Potassium Cyanide 187
Rare Earths—Separation 181
Rosin in Admixture with Coal-Tar Residues 190
Silicates, Decomposing by Hydrofluoric Acid 393
Silicon in Irons containing Vanadium or Molybdenum 178
Silver—Colorimetric 176
Sodium Paratungstate in Determinations by Loss 173
Sulfates—Jackson's Candle Turbidimeter 185
Sulfides in Alkaline Cyanides 185
Sulphides, Modified Qualitative Scheme 394
Sulfur in Coal 185
Sulfur in Iron and Steel 399
Sulfur in Roasted Blende 396
Sulfur in Roasted Pyrites 185
Sulfur Dioxide in Gelatines 191
Sulfur Monochloride for Decomposing Rare Earth Minerals 172
Tarry Matter in Mineral Cylinder Oils 189
Tellurium 399
Thallium—New Reaction 184
Thorium—Iodic Acid Separation 182
Tin and Antimony in White Metals 175
Tin in Canned Foods 191
Tin in Ores 175, 395
Tin Titrations 395
Titanium 180
Trimethylphenylammonium Carbonate 392
Tungsten Concentrates 182
Tungsten in Wolframite in Presence of Molybdenite 398

Turpentine, Rosin and Rosin Oil in 401
Vanadium in Iron and Steel—Colorimetric 180
Vanadium in Ores 182
Vanadium in Presence of Arsenic 182
Vanadium in Steels 180
Wulfenite 398
Zinc in Ores 397
Zinc Oxide in Glassware 172
Zirconia and Hydrofluoric Acid 183
Arch Ribs, Elastic, General Theory of, According to the Law of Least Work. By W. H. Burr 109
Arizpe, Sonora, Mex., Cyanide Plant at. By Edward L. Dufourcq 18
BARAGWANATH, J. G. Notes on the Geology of the Zaruma Mines, Ecuador 160
BLANCHARD, ARTHUR H. Graduate Courses in Highway Engineering at Columbia University 155
BOOK REVIEWS 106, 203, 312, 409
Addresses to Engineering Students. By Waddell & Harrington.... 312
Autobiography of John Fritz 314
Binary Alloys. By K. Bornemann 413
Building Stones and Clays; Their Origin, Characters and Examination. By E. C. Eckel 412
Contemporary Chemistry. By E. E. Fournier d'Albe 409
Cyanide Practice. By H. W. MacFarren 411
Direct and Alternating Current Manual. By Frederick Bedell 107
Dredges and Dredging. By Charles Prelini 411
Electric Central Station Distributing Systems. By H. B. Gear and P. F. Williams 108
Electric Traction and Transmission Engineering. By S. Sheldon and E. Hausman 106
Graphic Statics, Short Course in. By Cathcart and Chaffee 410
Higher Mathematics for Chemical Students. By J. R. Partington.. 409
Die Metallurgie des Wolframs. By Hans Mennicke 203
Metallurgy, A Brief Outline of the Modern Processes for Extracting the More Important Metals. By W. Borchers, Trans. by W. T. Hall and C. R. Hayward 204
Mining without Timbering. By R. B. Brinsmade 205
Petrographic Methods. By Ernst Deinschenk 410
Rock Minerals: Chemical and Physical Characters and Determination in Thin Sections. By Joseph P. Iddings 106
Short Course in the Testing of Electrical Machinery, for Non-Electrical Engineering Students. By J. H. Morecroft and F. W. Hehre 204
Stamp Milling. By A. DelMar 312
Steam Engine and Turbine. By Robert C. H. Heck 414
Theory and Practice of Technical Writing. By S. C. Earle 313
BURR, W. H. General Theory of Elastic Arch Ribs According to the Law of Least Work 109
Caisson Foundations for the Seamen's Church Institute Bldg. By M. Deutsch 358

Calorific Power, Relation of to Chemical Composition in Wood, Etc. By H. C. Sherman and C. G. Amend 30
CAMPBELL, W. Abstracts—Metallography and Alloys 402
Specifications for Certain Metals 105
CAMPBELL, W. and ALLEN, H. B. Heat Treatment of a Nickel Steel.... 72
Carbon Monoxide, Physiological Effects of. By Henry S. Munroe.... 340
Causes and Prevention of Coal Mine Explosions. By Edward K. Judd.. 65
China, Han-Yeh-Ping Iron and Coal Co. By Cho-Yang and Cheng-Fu Wang 165
Chispas Cyanide Plant, Arizpe, Sonora, Mex., by Edward L. Dufourcq.. 18
Coal Mine Explosions, Causes and Prevention of. By Edward K. Judd.. 65
Columbia University, Graduate Courses in Highway Engineering at. By Arthur H. Blanchard 155
Compressed Air, Pumping by. By H. T. Abrams 293
Copper Alloys, Volatilization of Gold and Silver from. By E. F. Kern and A. A. Heimrod 34
Cyanide Plant at Chispas Mine, Arizpe, Sonora, Mex. By Edward L. Dufourcq 18
DAVIS, WILLIAM H. Patents 209
DEUTSCH, M. Caisson Foundations for the Seamen's Church Institute Bldg. 358
Diamond Drilling, Speeds and Cost of, at Porcupine, Ont. By A. E. Hall 229
DUFOURCQ, EDWARD L. Chispas Cyanide Plant, Arizpe, Sonora, Mex. .. 18
DWIGHT, ARTHUR S. Efficiency of Ore Roasting 1
Efficiency in Ore Roasting. By Arthur S. Dwight 1
Electric Lamps, Modern Incandescent. By L. J. Lewinson 259
Electric Mining Applications. By E. A. Lof 317
Electrocyanide Process. The Parks. By John R. Parks..... 365
Explosions, Coal Mine, Causes and Prevention of. By Edward K. Judd.. 65
Force, Static and Dynamic, Relations in Machines. By W. Rautenstrauch 345
Foundations, Caisson, for Seamen's Church Institute Bldg. By M. Deutsch 358
General Theory of Elastic Arch Ribs, According to the Law of Least Work. By W. H. Burr 109
Geology of the Zaruma Mines, Ecuador. By J. G. Baragwanath..... 160
Gold and Silver, Volatilization of from their Copper Alloys. By E. F. Kern and A. A. Heimrod 34
Graduate Courses in Highway Engineering at Columbia University. By Arthur H. Blanchard 155
HALL, ALBERT E. Speeds and Cost of Diamond Drilling at Porcupine, Ont. 229
Han-Yeh-Ping Iron and Coal Co., China. By Cho-Yang and Cheng-Fu Wang 165
Heat Treatment of a Nickel Steel. By W. Campbell and H. B. Allen... 72
HEIMROD, A. A. and KERN, E. F. Volatilization of Gold and Silver from their Copper Alloys 34
Highway Enginering, Graduate Courses in at Columbia University, By Arthur H. Blanchard 155
JUDD, EDWARD K. Causes and Prevention of Coal Mine Explosions.... 65
KERN, E. F. and HEIMROD, A. A. Volatilization of Gold and Silver from their Copper Alloys 34

Lamps, Modern Incandescent Electric. By L. J. Lewinson 259
Least Work, General Theory of Elastic Arch Ribs, According to the Law of. By W. H. Burr 109
LEWINSON, LEONARD J. Modern Incandescent Electric Lamps..... 259
LOF, E. A. Electric Mining Applications 317
Machines, Static and Dynamic Force Relations in. By W. Rautenstrauch 345
Mechanical Conveyors, Study of. By L. DeG. Moss 136, 233
METALLOGRAPHY AND ALLOYS—ABSTRACTS. By W. Campbell..... 402
Alloys, Thermo-Electricity of, ... 405
Annealing of Metals 402
Brass, Heat Treatment of 404
Calcium, Binary Systems of 407
Cerium—Tin Alloys 403
Chrome Steels 406
Cobalt—Gold Alloys 403
Copper—Arsenic Alloys, Properties and Constitution of 404
Copper, Arsenical, Effect of Silver, Bismuth and Aluminum on..... 404
Copper—Silver Alloys, Conductivity and Hardness of 405
Corrosion, Tests of Steel by 406
Diffusion Alloys 402
Gold—Magnesium Alloys, Etc. 403
Iron—Carbon, The Stable System 406
Iron—Chrome Alloys 406
Iron—Copper—Nickel, Ternary System 405
Lithium Amalgams 403
Lithium, Behavior of, to Sodium, Potassium, Tin, Cadmium, and Magnesium 403
Lithium, Binary Systems of 407
Lead—Tin—Antimony Alloys, Constitution of 404, 405
Magnesium—Zinc—Cadmium, Ternary Alloys 405
Manganese—Aluminum—Copper Alloys, Magnetic Properties of 405
Manganese Oxide-Silica, System 406
Metallographic Notes 402
Molybdenum—Nickel and Manganese—Thallium Alloys 402
Osmondite, Formation of, in Hypo-eutectic Steels 406
Ovifak Iron; a Natural Steel 405
Phosphides of Iron 406
Phosphor Bronze 404
Phosphorus and Cobalt 404
Resistance of Steel to Deformation 406
Silver—Cadmium Alloys 403
Silver—Sodium Diagram 403
Slag in Thomas Steel, Nature of 406
Sodium, Binary Systems of 407
Steels, Color-etching of 405
Sulphur—Tellurium, Research on the System 403
Supplementary Bibliography 407
Technique of Metallography, Simplifications in 402
Tellurium, Alloys of, with Cadmium and Tin 403
Tin—Lead Alloys 404

Tin with Sulphur, Selenium and Tellurium, Thermal Diagram of.. 404
Metals, Specifications for certain 105
MINING DECISIONS, ABSTRACTS. By J. W. Thompson 84
Assumption of Risk 92
Care Required of Operator of Mine 89
Combination to Procure Title to Coal Lands 93
Contract for Purchase of Mining Claim 94
Contributory Negligence 90
Death of Miner by Suffocation—Assumption of Risk..... 92
Deed to Coal and Mining Privileges—Right to subadjacent Support.. 93
Duty of Mine Owner in case of Neglect of Mine Foreman 87
Duty of Mine Owner to keep Mine safe 91
Evidence—Report of Mine Superintendent 91
Excessive Mining Location 84
Failure to Discover Ore in Mining Discovery Shaft 84
Injury of Miner—Liability of Operator 89
Injury to Miner by falling Rock—Liability of Mine Operator..... 92
Injury of Miner—Measure of Damages 91
Injury to Miner—Safety Cages 89
Inspection of Mine—Liability of Mine Owner 90
Liability for Death of Person Rescuing Imperiled Miners 87
Liability of Mine Owner for Damages Caused by the Negligence of Another Miner 91
Liability of Mine Owner for Injury from Improper Ventilation.... 86
Life Insurance—Visiting Mine not a Forfeiture 94
Location of Mining Claim—Rights of Subsequent Locators..... 84
Mechanics' Lien on Mining Property—Filing 96
Mine Appliances or Fixtures—Sale on Execution 94
Nature of Oil and Gas Lease 86
Notice of Location 84, 85
Option on Mining Claims 94
Property subject to Miners' Liens 95
Statute making Mine Owners Liable for Negligence of Mine Foreman Unconstitutional 87
Taxation of Minerals 85
Taxation of Mining Claims—Description of Claim 95
Mining, Electric Applications. By E. A. Lof 317
Modern Incandescent Electric Lamps. By L. J. Lewinson 259
MOSS, LINCOLN DEG. Study of Mechanical Conveyors.....136, 233
MUNROE, PROFESSOR HENRY S. Physiological Effects of Carbon Monoxide 340
Nickel Steel, Heat Treatment of. By W. Campbell and H. B. Allen .. 72
Ore Roasting, Efficiency of. By Arthur S. Dwight 1
Parks Electrocyanide Process. By John R. Parks 365
PARKS, JOHN R. The Parks Electrocyanide Process 365
Patents. By William H. Davis 209
Physiological Effects of Carbon Monoxide. By Henry S. Munroe..... 340
Porcupine, Ont., Speeds and Cost of Diamond Drilling at. By A. E. Hall 229
Pumping by Compressed Air. By H. T. Abrams 293

RAUTENSTRAUCH, W. Static and Dynamic Force Relations in Machines. 345
Relation of Chemical Composition to Calorific Power in Wood, Etc. By H. C. Sherman and C. G. Amend 30
Roasting, Efficiency of Ore. By Arthur S. Dwight 1
Seamen's Church Institute Bldg., Caisson Foundations for. By M. Deutsch 358
SHERMAN, H. C. and AMEND, C. G. Relation of Chemical Composition to Calorific Power in Wood, Etc. 30
Speeds and Cost of Diamond Drilling at Porcupine, Ont. By A. E. Hall. 229
Static and Dynamic Force Relations in Machines. By W. Rautenstrauch 345
Study of Mechanical Conveyors. By L. DeG. Moss 136, 233
Volatilization of Gold and Silver from their Copper Alloys. By E. F. Kern and A. A. Heimrod 34
WALLER, E. Abstracts—Analytical Chemistry 172, 392
WANG, CHENG-FU and YANG, CHO-. Han-Yeh-Ping Iron and Coal Co., China 165
YANG, CHO- and WANG, CHENG-FU. Han-Yeh-Ping Iron and Coal Co., China 165

VOL. XXXIII. NO. 1. NOVEMBER, 1911

THE SCHOOL OF MINES QUARTERLY.

A

JOURNAL OF APPLIED SCIENCE.

CONTENTS

	PAGE
EFFICIENCY IN ORE ROASTING. By *Arthur S. Dwight*.....	1
CHISPAS CYANIDE PLANT, ARIZPE, SONORA, MEXICO. By *Edward L. Dufourcq*.....	18
RELATION OF CHEMICAL COMPOSITION TO CALORIFIC POWER IN WOOD, ETC. By *H. C. Sherman and C. G. Amend*.....	30
VOLATILIZATION OF GOLD AND SILVER FROM THEIR COPPER ALLOYS. By *Edward F. Kern and A. A. Heimrod*.....	34
CAUSES AND PREVENTION OF COAL MINE EXPLOSIONS. By *Edward K. Judd*.....	65
HEAT TREATMENT OF A NICKEL STEEL. By *W. Campbell and H. B. Allen*	72
ABSTRACTS OF RECENT MINING DECISIONS.....	84
SUMMER ACTIVITIES	97
PERSONAL ITEMS	103
BOOK REVIEWS	106

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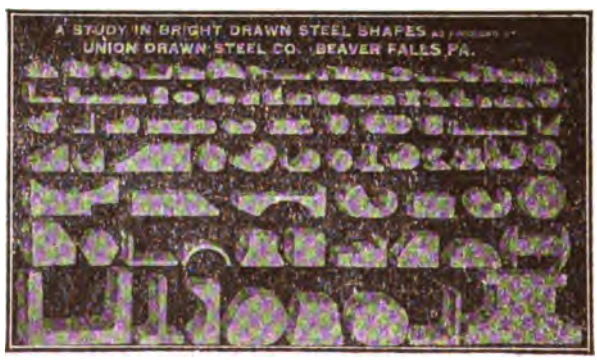

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THE
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VOL. XXXIII. NOVEMBER, 1911. No. 1.

EFFICIENCY IN ORE ROASTING.

BY ARTHUR S. DWIGHT.[1]

This is a study of the relative efficiencies of some of the more important desulphurizing methods used in the metallurgy of lead and copper, to prepare the ore for smelting. It is proposed to show how these efficiency factors have served to shape the development of the roasting process from the crude and inefficient method of burning ore in heaps, through the multifarious forms of reverberatory furnaces, up to the recently devised systems of blast-roasting.

The exact purpose of the roasting operation is not always the same; thus in roasting lead ores, it is usually desired to eliminate as much of the sulphur as possible, in order to avoid making, in the smelting operation, an undue quantity of matte, a troublesome by-product which locks up valuable metals that ought to have been set free in the metallic product, and thus calls for an extra treatment to recover these metals. With copper ores, on the contrary, a matte is the logical product of the first smelting, and hence it is customary to leave a definite remnant of the sulphur to "cover" the copper and prevent unnecessary losses of metal in the slag. Again, on account of the greater fusibility of lead ores, and the ease with which lead may be volatilized and lost, a more careful heat regulation is necessary in their treatment, and this tends not only to lower the efficiency factor of a given type of furnace, but definitely to debar certain types from service with lead ores. However, these variations are no greater than

[1] Consulting mining engineer, 25 Broad St., New York City; E.M., Columbia, 1885.

those which may be caused by the varying skill shown in design and operation, and we may, therefore, neglect them for the purposes of the present discussion. It is sufficient to say that the relative efficiency factors here deduced will be approximately true for either lead or copper ores, though special metallurgical requirements may dominate the choice of apparatus.

The roasting capacities of different types of furnaces depend upon the same general principles, viz: (1) the quantity of ore that can, within a given period of time, be effectively exposed to the oxidizing influence of the air, and (2) the rapidity with which the oxidizing action can be effected. Large area of hearth and frequency of stirring are the means generally employed for increasing the quantity of ore thus exposed, while careful control of the heat and air supply, fine crushing, and intelligent regulation of the chemical composition of the ore mixture contribute to the speed of oxidation.

Dr. E. D. Peters[2] thus summarizes the fundamental requisites for rapid and efficient roasting: "(a) small ore particles. (b) A vigorous air current, which shall constantly bring fresh oxygen to the surface of the ore, and shall also sweep away the residual nitrogen of the spent air and the sulphur-dioxide gas which would dilute the atmosphere in contact with the ore so that its oxidizing power would lose its efficiency. (c) Frequent stirring of the ore, for, after the removal (approximately) of the first atom of sulphur from the pyrite—which atom is driven off by heat alone — the oxidation of the ore proceeds with discouraging slowness as soon as the upper layer of sulphide particles has become partially oxidized."

To determine the relative efficiency of different types of furnaces requires us to go a step further than the mere capacity of the furnace, as the question of fuel, labor, repairs, interest charges on the plant and on the value of metals locked up in the process, and finally the character of the product, have an important bearing. Many of these factors are variables, even when considering the same type of furnace under different conditions; hence it is difficult to select one factor which will tell the whole story.

[2] "Some Future Problems in the Metallurgy of Copper," *Mineral Industry*, Vol. XVII, 1909, p. 282.

Taking everything into consideration, the best factor we have is the *number of pounds of ore treated per day on a square foot of the furnace hearth.* This factor involves the time-element, which must necessarily enter largely into the make-up of most of the other factors, and it also involves the factor of size which gives a certain measure of the size of the furnaces and the buildings required for handling a given tonnage, and this, in turn, is a rough index of the first cost of the plant. As regards the important question of fuel, a "free-burning" ore will, when charged into the hot furnace, take fire and burn without any external fuel being required; and, although certain styles of furnaces are more suitable for propagating and maintaining this reaction than others, the actual efficiency factor will not be materially different, whether all of the necessary heat is supplied by the sulphur in the ore itself or by coal burned on the grate, though the *cost of roasting* will probably be much higher when the external fuel is used. Again, the character of the roasted product, its fineness or coarseness, is a matter of great importance. If the blast furnace is to do its best work, it must be given material reasonably coarse and preferably cold. The reverberatory smelting furnace, on the contrary, does best when the charge is made up of hot fines. The smelter of copper has the choice of both types of smelting furnaces, but the smelter of lead is limited to the blast furnace, and hence is likely to be much more solicitous that his roasting operations shall yield him a coarse product. Briquetting the fine roasted ore has helped him to secure this desideratum, but this, in turn, has involved a separate and expensive operation, with results not entirely satisfactory, so that briquetting is adopted only with reluctance. Other things being equal, that furnace which will turn out a product in coarse form is entitled to a long credit mark, and this may surpass in practical importance its efficiency factor.

We may divide those roasting methods which are pertinent to this discussion into four principal classes:

I. Roast heaps and stalls.

II. Reverberatory roasters:

(1) Hand-stirred furnaces. (Hand-reverberatory.)

(2) Mechanically-stirred furnaces of diverse styles.

(3) Revolving cylinders. (Brückner, etc.)

III. Blast-roasting in pots by the Huntington-Heberlein, Savelsberg, Carmichael-Bradford and McMurty-Rogers methods.

IV. Blast-roasting of thin layers (Dwight-Lloyd System).

1. Intermittent down-draft pans.
2. Continuous-sintering machines.

It is not intended to describe in detail the different methods and furnaces here referred to, but merely to indicate those points which relate to this discussion, as well as those which are necessary to show the basis for calculating the efficiency factor.

I.—The Roast Heap.

This is the most primitive form of roasting appliance, requiring practically no plant but the area of ground upon which the coarsely broken ore is piled on a layer of cordwood to kindle it, and covered with a layer of finer ore to control the degree and direction of the draft. Some of the irregularities of heap-roasting may be prevented by conducting the operation in masonry stalls with paved floors containing air ducts, and sometimes a system of flues connected with a chimney to assist the draft and dispose of the noxious sulphur gases, but the principles involved and the efficiency factors are about the same. The bulk of the ore in the pile must be coarse, to permit the free passage of the gases. From the very nature of the operation, no rabbling or stirring is possible during the roasting, for which reason the results are at best extremely irregular. When a finished heap is discharged, streaks of imperfectly roasted material will be found which must be sorted out and subjected to a second fire and sometimes even a third. This sorting and re-treatment may be considered as an archaic form of stirring or rabbling. Sometimes the same result is obtained more efficiently by successively stripping off the outside of the heap where the ore naturally roasts more quickly and effectively than in the interior. By thus scraping off the outer layers as fast as they are finished, the inner portions are successively given this same advantage until the entire pile is consumed.

Cheap and plentiful labor, though not necessarily skilled, is an essential when so much handling and re-handling is called

for, but granted this, the cost of roasting a ton of ore in heaps may be as low as by the most modern and approved methods in spite of its low efficiency factor. The physical character of the product is generally favorable for the blast furnace, being coarse and spongy. Heap-roasting is frequently found to be an effective aid in getting a new metallurgical enterprise started quickly, on account of the absence of complicated plant, but is seldom used as a permanent method, at least under competitive conditions, because labor is usually not sufficiently cheap and plentiful. Losses by wind and by the leaching of soluble metallic salts by rain may be serious, particularly with copper ores. The great objection to heap-roasting, however, is that it is so slow that the interest charge on the capital locked up in the ore undergoing treatment becomes a serious item of expense. In other words, its efficiency factor is low. For this discussion, we may take the efficiency factor at 5 to 20 lb. per square foot of hearth area.

To illustrate this most primitive method, we may select the practice of heap roasting at Goslar, Lower Harz, Germany.[3] The heaps are 35 to 40 ft. square at bottom and 12 ft. square at top, with a thickness of about 6 ft., resting on a layer of cordwood and faggots 16 to 20 in. thick, the whole being heavily blanketed with a thick layer of fines. The contents is about 400 tons. First roasting lasts six to seven months, after which the unroasted portion is sorted out and subjected to a second and even third fire of shorter duration. The "hearth area" is regarded as equivalent to the area of the average horizontal section of the pile, here taken at 872 sq. ft. Conceding that with favorable conditions and skillful manipulation the entire contents of the pile might be satisfactorily roasted in the first period of six months duration, the efficiency factor will be:

$$E=\frac{800{,}000 \text{ lb.}}{872 \text{ sq. ft.} \times 180 \text{ days}}=5.1 \text{ lb. per day per sq. ft. of hearth.}$$

There are examples of more efficient heap-roasting than this, for the Gosler practice prolongs the roasting period to what seems an unnecessary degree. The explanation is probably found in

[3] Collins, "Metallurgy of Lead," second edition, London, 1910, p. 70.

the fact that the recovery of part of the sublimed sulphur was attempted. This result would be promoted by heavy blanketing of the exterior by means of the layer of fines and this would naturally make the roasting slow.

In commencing our series of efficiency factors with the heap roast at 5 lb. per sq. ft. per day, we do so with the further explanation that in other localities heap-roasting has been brought up to a relatively high degree of perfection, so that the efficiency factor may easily run as high as 20 lb. per sq. ft. per day, and cases are on record where it has approached very close to the efficiency factor of the hand roasters, next to be described.

II.—Reverberatory Roasters.

The prototype of all the furnaces of this class is the German long hearth reverberatory furnace, the so-called *Fortschaufelungsofen*. This has a long rectangular hearth covered by a low, flat, reverberatory arch, with a fire-box at one end and an exit-flue at the other, and numerous working doors on both sides to admit the long handled tools for stirring or rabbling. The ore is fed to the furnace near the exit-flue, spread out in a thin layer on the hearth, frequently stirred to expose new surfaces for oxidation, and gradually moved by the rakes and paddles toward the fire bridge when it is exposed to the highest desirable heat just before being discharged. The labor of rabbling is severe in the extreme, and it was natural that efforts should have been made to relieve the workmen and accomplish the stirring by mechanical means. This gave rise to a great variety of mechanical roasting furnaces, of all shapes and sizes, and employing widely different mechanical devices for moving and stirring the ore, but the principle underlying all of the furnaces of this general type, both hand- and mechanically-operated, is the same.

From this point of view it will be interesting to observe how the efficiency factors of these highly developed forms of automatic roasting furnaces compare with one another, and with their own prototype, the hand roaster.

Hand-Stirred Reverberatory.—The general features of this furnace have been already described. The earlier furnaces were smaller than those of later construction, but a length

of 65 ft. and a width of 15 ft. were found to be the practical limits of hearth area, and very few improvements have been made in this furnace during the past 20 years. By reason of the greater flexibility of hand-rabbling, this furnace has been found peculiarly suited to roasting lead ores, and in spite of all the efforts that have been made, in the interest of economy of labor, to adapt the many automatic furnaces to this service, the hand roaster has remained, almost to this day, the one trusted and reliable roaster for the lead metallurgist, that is able to meet any situation that may arise. It is costly to operate, both as to fuel and labor, and occupies an enormous area in proportion to its output (i. e. its roasting efficiency is low) but it is *reliable*. Another reason for its long popularity is the favorable character of its product. By care in making up the charges, and skill in regulating the heat at the finish, the particles of roasted material can be discharged in a sort of nodulized condition which is considered reasonably favorable for the blast furnaces.

One modification of this furnace which was extensively used in the West for some years was the slagging roaster, which was originally used at the Omaha & Grant smelter, Denver, Colo. This consisted of a combination of a reverberatory melting hearth next to the fire bridge, and a long flat hearth in which the preliminary roasting was effected by the waste gases from the fusing-box. The ore, during roasting, was gradually moved toward the fusing-box, and finally scraped in and melted in batches, forming a slag which was drawn into pots, and, when cold, was sent to the blast furnace. This product was highly appreciated for its coarse mechanical condition, but the practice was finally abandoned on account of high metal losses. It was an interesting development, however, as an illustration of the price the lead metallurgist was willing to pay for a coarse smelting material.

The capacity of a typical hand roaster with a hearth 60x15= 900 sq. ft. was about 12 tons per 24 hours or 26.6 lb. per sq. ft. per day. Collins[4] gives the following comparative factors of efficiency, in pounds per sq. ft. per day: Przibram, Bohemia, 31; Laurium, Greece, 28; Port Adelaide, Australia, 12; Cockle Creek, Australia, 29; Colorado Smelting Co., Pueblo, Colo. (Eilers plant),

[4] Op. Cit., "Metallurgy of Lead," p. 85.

37; St. Joseph Lead Co., Bonne Terre, Mo., 30; No. 3, Monterey, Mexico, 24; Murray plant, Utah, 24. We may, therefore, take the efficiency factors for hand roasters as between 24 and 35 lb. per sq. ft. per day.

Mechanically Stirred Reverberatories.— The development of the automatic roaster from the hand-reverberatory has been already referred to. The line of improvement follows almost exclusively the one idea of more frequent stirring of the ore. Again to quote Dr. Peters,[5] "What I wish to point out in this immediate connection is that this question of a more frequent stirring of the ore was the one glaring fault in our roasting practice of 30 years ago, and, that, really, almost the entire energy and ingenuity of the inventors and users of automatic furnaces were focussed upon this single point. All the wealth of science and of modern mechanical skill has been brought to bear upon the exposing of fresh particles of ore to heat and oxygen as frequently as is needful, and has been successful."

Mechanically stirred furnaces had been used many years ago in England, principally for roasting tin ores, but apparently at that time they did not fit into the requirements of the copper and lead business, or, if so, it was to a very limited extent. The Parkes, original MacDougall, and Brunton furnaces were invented at that time, and have served as the prototypes of some of the most successful of the modern automatic furnaces. The O'Harra was developed in the early days of Butte, Mont., and became the prototype of another series of adaptations. In fact, there was a voluminous crop of patent furnaces, and almost every metallurgist of renown had one of his own. The rivalry was intense, and furnished an interesting and probably a necessary chapter in the history of American metallurgical practice. Some of these furnaces had commendable features, but the chief praise that could be given to most of them was that they were "different." Of the many that once bewildered the metallurgist, a few that had the requisite combination of merits have survived. These merits are simplicity, durability and low repairs, low first cost and satisfactory efficiency. A glance at the table of relative efficiencies of these automatic furnaces will show that there was an

[5] Peters, Op. Cit., *Mineral Industry*, Vol. XVII, 1909, p. 283.

exceedingly small range of variation among the different styles, considering the extent of the effort that was made for improvement. The following table gives representative figures of efficiencies (pounds per sq. ft. per day) for furnaces of different styles operating under widely different conditions, in both lead and copper work.

EFFICIENCY FACTORS OF VARIOUS STYLES OF MECHANICAL ROASTERS

	Collins [6]	*Kern* [7]
Brown-Allen-O'Hara	34	74
O'Harra, Butte, Mont.	—	33- 36
Brown horse-shoe	48	35- 38
Brown horse-shoe (treating matte)	38	
Pearce turret (treating matte)	43- 55	60- 66
Ropp straight-line, Port Pirie, Australia	95	
Ropp straight-line, Hanauer, Utah	33	
Ropp straight-line, general conditions	60	67- 77
Keller-Cole-Gaylord, Germania, Utah	104	44- 48
Heberlein, light roast	150	
Heberlein, Sullivan, B. C.	169 (?)	
Godfrey, Murray, Utah	54- 80	
Godfrey, Australia	141	
Wethey, original		60- 65
Wethey, improved		65- 70
McDougall, Evans-Kleptko, Anaconda, Mont.		84- 86
Edwards		140-152
Herreshoff-McDougall		30- 35

It is evident, therefore, that we may safely claim for furnaces of this class an efficiency varying from 33 to 75 lb. per sq. ft. per day for ordinary conditions, and as high as 150 lb. per sq. ft. per day when but little sulphur elimination is required. This latter figure may properly be used in making comparisons with the blast-roasting processes.

Revolving Cylinder Roasters. — As exemplified by the Brückner cylinders, this type of roaster was very popular at one time both in lead and copper work, but is now almost entirely abandoned. The high proportion of flue-dust made by the Brückners was one of the principal causes of the gradual abandonment of this method of roasting. It is to be regarded essentially as a revolving reverberatory furnace in which the inner lining of the cylinder becomes successively the hearth and the reverberatory arch of the furnace, and the exposure of fresh surfaces of ore is accomplished by the changing talus of the material as the cylinder slowly re-

[6] Collins, Op. Cit., page 90.
[7] E. F. Kern, (by permission) Lecture Notes, 1910, (unpublished).

volves. The area of the hearth may be taken for practical purposes as equivalent to the horizontal projection of the cylinder.

At the Germania plant, in Utah, the Brückner practice attained its highest perfection. Here the cylinders were 22 ft. long by 8½ ft. diameter, and treated 12 tons per day. Equivalent area of hearth was 187 sq. ft., making the efficiency factor 128 lb. per sq. ft. per day.

To summarize the characteristics of the automatic roasters of the reverberatory type, we may say that they show a distinct gain in efficiency over the hand-stirred type, though the gain is not so marked as one would expect. The truth is that a practical limit is soon reached as to the amount of sulphur that can be burned on a square foot of hearth area in a given time without causing the sulphides to melt and interfere with the further progress of the reaction, or clog the stirring mechanism. This practical limit probably corresponds closely to the efficiency factor of 75 lb. per sq. ft. per day, though in certain special cases where the ore contains less sulphur, or from its nature will bear a higher temperature without fusing, or where there are special provisions for moderating the resulting temperatures, (as in some of the improved types of MacDougal furnace, notably the Wedge furnace), the efficiency factors may be carried as high as 175 lb. per sq. ft. per day.

III.—Blast Roasting in Pots.

The last paragraph called attention to the great stumbling-block in the way of increasing the efficiency of the mechanically stirred reverberatory, arising from the too rapid generation of heat by the mass-action, whereby the sulphide is caused to melt or "matte." The next important advance in the art of roasting resulted from discovering how to control and check this mass-action. It was accomplished by mixing the grains of sulphide to be roasted with grains of an inert or refractory material in such proportion that the particles of sulphide are held apart from one another by refractory grains. If a mass of material so constituted be ignited at one surface and a blast of air forced through it, the sulphide particles will successively shrivel up in the fierce oxidizing blast, and the reaction will propagate itself through the

mass in the direction of the air current. An additional advantage may be secured if the isolating grains are of such composition that, while they remain inert during the roasting period, they will, at the temperature developed by the cumulative mass-action, combine with the newly formed metallic oxides to form a reasonably fusible slag or sinter. This is the action as we now understand it, but at first the reason for the extraordinary rapidity of the roasting was not so evident. Instead of recognizing it as a purely mechanical effect, Messrs. Huntington and Heberlein, who first developed this process, were misled into the belief that some powerful chemical reagent was at work, and that the burnt lime they employed as an isolating ingredient of the charge was the vehicle for nascent oxygen in the form of CaO_2. So sure were they of this fact that they based their patents in all the important countries of the world upon this reaction, and the theory was generaly accepted as true until research demonstrated that it was untenable. Even then it was generally held that the lime had some important, though mysterious chemical effect, and the process was still called "Lime Roasting." It has now been demonstrated that many other substances will answer as well, if not better than lime, and "Blast Roasting" is now generally accepted as the more logical and precise designation.

Although Messrs. Huntington and Heberlein did not realize at first the true cause of their success, they are none the less entitled to great credit for working out a successful process which constituted a great step in advance, and removed some of the most perplexing difficulties of the lead metallurgist. They made it possible to roast fusible galena ores cheaply, efficiently and with diminished metal losses, yielding a product of remarkably favorable physical character, which greatly promoted the efficiency of the subsequent smelting. Although the form of apparatus they employed (blast-pots) was crude and primitive, the efficiency factor was far higher than had ever been previously attained by what Dr. Peters has referred to as the "wealth of science and modern mechanical skill," as applied to the roasting problem.

The procedure originally proposed by Huntington and Heberlein required a pre-roasting of the charge before treatment in the pots, but as experience was gained in manipulation it was found

possible, under some conditions, to dispense with the pre-roast and subject the charge to direct converting. The variations in the process devised by Carmichael and Bradford in Australia, and by Savelsberg in Germany, dispensed with pre-roasting. The first named investigators noticed that sulphate of lime was quickly formed in the converter, and reasoned that it might as well be introduced at the start in place of the burnt lime. Savelsberg carried this reasoning still further, and substituted crushed limestone. Patents were granted to all of these investigators, and their processes have found extensive utilization. But the efficacy of the lime was unquestionably overestimated, and its true function misunderstood.

The original Huntington-Heberlein pot was small and deep. As the roasting and sintering action propagates itself progressively upwards, the time needed to complete a charge depends more upon the distance which this slowly moving zone has to travel than upon the actual weight of material treated. The present writer has shown in a previous article [8] that, on account of this progressive sintering, most of the space in a converter is utilized for storing rather than for treating its contents. It was to be expected, therefore, that the improved forms of pots developed by later experience should be larger and shallower. As the depth was diminished, however, the proportion of unsintered fines tended to increase, and this tendency limited the improvement in that direction. The reason for this is found in the fact that the gases escaping at the top of the pot keep the loose particles on the top surface in constant agitation and thus prevent the close brooding contact which is necessary to secure complete sintering. With a modern 10-ton pot, 9 ft. diameter by 4 ft. deep, the proportion of fines will usually amount to about 10 percent., sometimes as high as 25 percent.

The pot discharges its product in the form of a huge cake of sintered material, which requires considerable hand labor to break into suitable size for smelting, even after a preliminary shattering of the cake has been accomplished by dropping it from a height. On account of its coarseness and its more or less cellular structure, the pot product is highly prized as material for the blast furnace.

[8] *Eng. and Min. Journ.*, March 28, 1908.

The initial sulphur in a properly proportioned blast-roasting charge should not be higher than about 18 percent. if the most efficient results are to be obtained, and this fact should be kept in mind in comparing the efficiency factors with those of the older types of roasters. As already stated, the most favorable efficiency factor of the automatic reverberatory roasters (175 lb. per sq. ft. per day) may properly be used in making comparisons with blast roasting results. The grate in the bottom of the pot, through which the air enters the charge, is much smaller than the diameter of the pot, and as there is a very complete diffusion of the air through the charge it seems only proper in calculating the equivalent hearth area of a pot, to use the area of the average horizontal section, just as was done in the case of the roast heaps, previously discussed.

The published figures of costs and capacities of the early Huntington-Heberlein work show some serious discrepancies, and are usually much more favorable than the known work of present-day plants using the same process and presumably improved practice. The following cases have been selected as fairly showing the range of capacities:

Huntington-Heberlein.[9] — Converters hemispherical 9 ft. diameter by 4 ft. deep, holding 8 tons material. Grate 6 ft. diameter, but assume equivalent hearth area (7½ ft. diameter) 44 sq. ft. Time of blowing 12 to 18 hours. Allow 1½ turns per day. Hence, capacity is 12 tons per day per converter, and efficiency factor is 545 lb. per sq. ft. hearth area per day.

Huntington-Heberlein, Trail, B. C.[10] — Converters, 8 ft. 8 in. diameter by 4 ft. deep, hold 12 tons. Equivalent hearth area 44 sq. ft. Capacity 15 tons per day per converter. Efficiency factor 680 lb. per sq. ft. per day.

Savelsberg.[11] —Pots, 8 ft. diameter, contain 8 tons; equivalent hearth area, 38.5 sq. ft. Time of blowing 18 to 20 hours, say 1¼ turns per day, or 10 tons per day per converter. Efficiency factor is 520 lb. per sq. ft. per day.

McMurty-Rogers.[12] — Pots, 8 ft. 8 in. diameter by 4½ ft. deep; take 8-ton charge. Equivalent hearth area, 44 sq. ft. Time of blowing 12 hours. Assume 1¾ turns per day; capacity is 14 tons per day per converter. Efficiency factor is 636 lb. per sq. ft. hearth per day.

Roasting-boxes, Bingham Junction, Utah.[13] — Hearth area 36 sq. ft. Capacity 16 tons per day. Efficiency factor 890 lb. per sq. ft. per day. (Considerable percentage of fines requiring retreatment.)

From the above figures we draw the conclusion that the average

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[9] Ingalls —"Lime Roasting of Galena." *Trans.* A. I. M. E., Vol. XXXVII, 1906.

[10] Hofman — "Recent Progress in Blast Roasting." *Trans.* A. I. M. E., Vol. XLI, 1910.

[11] Ingalls, "The Savelsberg Process," *Eng. and Min. Journ.*, Dec. 9, 1905.

[12] *Mineral Industry*, XVI, 1908, p. 334.

[13] Hofman. Op. Cit.

efficiency factors for pot roasting lie betwen 500 and 900 lb. per sq. ft. per day, with average probably close to 600 lb. This is indeed a remarkable gain in efficiency over the best work of the reverberatory furnace under equivalent conditions of sulphur elimination, and fully justifies the verdict pronounced by Prof. Joseph W. Richards in 1908[14] that pot roasting was "the most radical improvement in recent years in the metallurgy of lead."

IV.—Blast Roasting of Thin Layers.

Attention has here been called to two of the weak points of the pot-roasting procedure caused by the use of the thick layer, viz: The ineffective utilization of the space in the converter, and the massiveness of the cake produced. Very early in his efforts to devise a means for adapting the blast-roasting idea to the treatment of copper fines, the present writer came to realize the economic importance of finding a means for *completely sintering a thin layer,* in order to eliminate these two objectionable features. When a thin layer, say 4 in. thick, was treated on an ordinary grate with up-draft, the bottom 2 in. would be sintered and the upper 2 in. would remain pulverulent. This was the net result of the efforts of many who experimented in this line. Finally it was found that by applying a mechanical means for holding quiescent the particles which had usually remained unsintered and maintaining them in close contact with one another, a complete agglomeration could be effected. It was found that this could be accomplished by restraining screens with up-draft or side-draft, but the simplest way was to constitute the supporting grate the restraining screen by reversing the usual direction of the air-blast and passing the air downwards. This completely solved the problem. It had the beneficial effect also of automatically preventing channelling and the formation of blow-holes. The resulting gain in efficiency was remarkable. When the principle was applied to shallow down-draft pans operating intermittently, the results were much superior to the best work of the pots, the product was more porous and easily broken to the desired size, and, on account of the lesser mass-action and greater heat radiation of the thinner layer, a wider range of chemical composition was permissible. Further

[14] "Metallurgical Calculations," Vol. III, page 560.

study along these lines pointed out the way to making it a *continous* process, and this has still further increased the efficiency factor, so that the final result achieved shows almost as great an improvement over the pots as these had previously shown over the best automatic reverberatories.

Intermittent, Dwight-Lloyd Down-Draft Pans.— The intermittent procedure has been tried generally in an experimental way or as a preliminary to the adoption of the continuous process, by which the former has almost invariably been superseded. Accounts have been published of two sintering plants that were designed to treat thin layers intermittently, following the procedure described in the fundamental patent of Dwight and Lloyd.

One of these plants was that built at Cerro de Pasco, Peru, in 1907, under an arrangement with the owners of the Dwight-Lloyd patents, but following independent designs. The description [15] is incomplete and the figures not entirely consistent, which is probably explained by their being taken from a Peruvian government report. The installation appears to have been not very satisfactory, and did not operate long. The reasons are sufficiently clear.

The other plant [16] was built at Port Pirie, Australia, by the Broken Hill Proprietary Company, to be used in connection with the Carmichael-Bradford gypsum process. It has since been superseded by the Dwight-Lloyd continuous process without gypsum. Its efficiency factor was about 1800 lb. per sq. ft. per day.

The principal reason why the efficiency factor of the intermittent apparatus is lower than that of the continuous is because of the time lost in the former by the operations of charging, igniting and discharging, and the difficulty of knowing just how the process is progressing and when it is finished. This loss of efficiency has been found to be 30 to 50 per cent. Under reasonably favorable conditions the writer has found the efficiency factor of the down-draft intermittent method to be 1000 to 2000 lb. per sq. ft. of hearth per day.

Continuous Process.— No attempt will here be made to describe

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[15] *Min. and Sci. Press,* Jan. 30, 1909.
[16] Collins, Op. Cit.

the details of this method, but the reader is referred to some of the several published descriptions.[17] It is sufficient to say that in the method now generally used a thin layer of the ore is fed on a continuously moving conveyor, with grated bottom, travelling over an air-tight box connected with an exhaust fan, which causes air currents to be drawn down through the layer of ore and the grate upon which it rests. Immediately after the stream of ore emerges from the feed hopper its level top surface is ignited by momentarily passing under an igniting device. Thenceforward the heat necessary for the propagation of the roasting and sintering reaction is derived from the internal combustion of the sul-

RESUMÉ OF EFFICIENCY FACTORS

	Efficiency factor Pounds ore treated per day per sq. ft. hearth.	Character of product for blast-furnace smelting.
I. Roast heaps and stalls	5 to 20 lb.	Good
II. Reverberatory Roasters:		
(1) Hand roaster	24 to 35	Fair
(2) Mechanically stirred:		
average conditions	33 to 75	Too fine
special conditions	150	" "
(3) Revolving cylinders	128	" "
III. Blast roasting in pots	500 to 900	Excellent
average	600	"
IV. Blast roasting of thin layers, (Dwight-Lloyd system):		
(1) Intermittent down-draft pans...	1000 to 2000	"
(2) Continuous sintering machines..	2200 to 3000	Ideal

phur or other combustible ingredient of the charge. The speed of travel is so regulated that when a given portion of ore has travelled across the suction box to the point of discharge, the zone of sintering action has progressed down to the grates, and the operation is complete. The fact that all the adjustments are reduced to mechanical terms makes it possible to keep the closest

[17] See: A. S. Dwight, *Eng. and Min. Jour.*, March 28, 1908; *Mineral Industry*, Vol. XVI, p. 381; H. O. Hofman, "Recent Progress in Blast Roasting," *Trans.* A. I. M. E., Vol. XLI, 1910, p. 739; F. M. Smith, *Journal* of the Association of Engineering Societies, Vol. XLV, July, 1910; Jas. Gayley, *Bulletin* A. I. M. E., August, 1911.

watch on each condition independently and quickly to correct any departure from normal.

The hearth area for purposes of calculation is the horizontal area of the layer undergoing treatment, which is the same as the top area of the suction box. Assuming average figures for specific gravity of ores and for sulphur contents, a standard Dwight-Lloyd sintering unit (type E, 42 × 264 in.), with an effective grate area of 77 sq. ft., will easily treat 85 tons per day of average suitable material without a pre-roast, which is equivalent to an efficiency factor of 2200 lb. per sq. ft. hearth area per day; while with a charge corresponding to those with which the best results are obtained in the pots, the capacity runs as high as 120 tons per day, equivalent to an efficiency factor of 3000 lb. per sq. ft. per day.

CHISPAS CYANIDE PLANT, ARIZPE, SONORA, MEXICO.

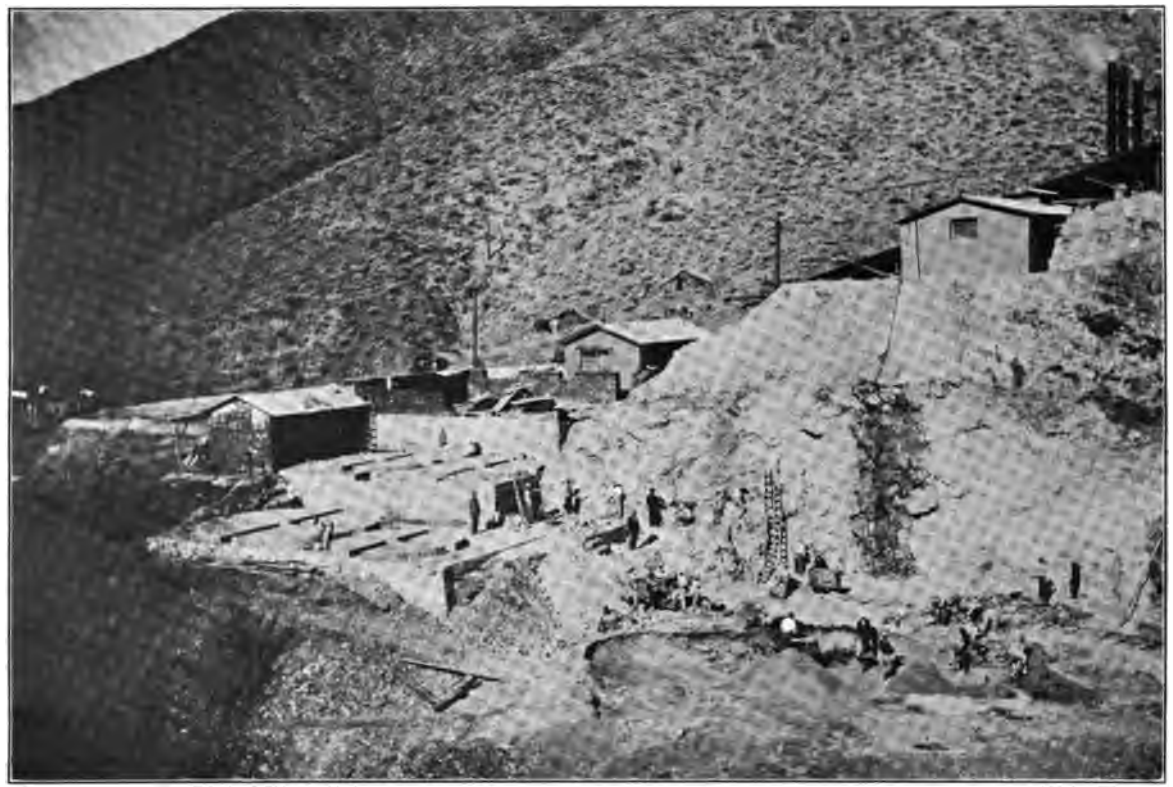
BY EDWARD L. DUFOURCQ.[1]

The Chispas mine, situated in the Arizpe district of Sonora, Mexico, and owned by the Minas Pedrazzini Gold and Silver Mining Company, was worked from 1891 until 1900 for its production of high-grade shipping ore. The period from 1900 until 1903 was occupied in running a drainage tunnel under the former workings. At the time at which I assumed the management of the property in 1904, the veins had been cut by this tunnel and, in addition to some high-grade ore, considerable amounts of lower-grade ore had been discovered; a five-stamp mill had been erected with one concentrating table and two amalgamating pans.

This mill was enlarged as more ore was developed, until in 1910 it consisted of 20 stamps, three Wilfley Tables, four 5-ft. pans and two settlers. During part of this period a portion of the slimes that did not pass through the pans in the ordinary amalgamation treatment had been hoisted from the settling ponds after drying and oxidizing, and had been treated in the pans by being mixed with the current coarse tailings from the concentrating tables. This re-treatment of slimes was costly on account of the second handling, the extraction was unsatisfactory, and moreover the total current production of slimes could not be treated in the plant. The extraction in the mill averaged 58 to 60 percent. by concentration and the pan-amalgamation then extracted from 55 to 60 percent. of the contents of the table tailings, the total extraction therefore, varying from 71 to 84 percent.

In 1908 and 1909, experiments were made with cyanide and it was decided in 1910 to install this process, both because of the higher extraction possible and because of the much lower cost of treatment than by pan amalgamation. In this paper I shall outline the construction and detail of the cyanide plant and in a

[1] Consulting mining engineer, Produce Exchange Bldg., New York City; E. M., Columbia, 1892.



Foundations of Cyanide Plant, January 15th, 1911

later paper shall be able to give the results of operations, the extraction effected and the costs of the process.

Some description of the geology of the ore deposits was given in the *Engineering and Mining Journal* of December 3, 1910, so that for the purposes of this paper it will be sufficient to mention that the ore consists of a rather clean quartz carrying the silver in the upper levels of the mine as chloride, and in the lower levels as argentite, polybasite and stephanite. The gold is free in the upper levels and throughout almost the whole vein is carried in about the proportion of 1 oz. of gold to 130 oz. of silver. The average of a considerable number of partial analyses of the ore gave the following result: Insoluble, 80.6; iron, 2.4; lime, 1.6; sulphur, 1.6; antimony, 0.3 percent.; zinc, trace; lead and copper, none.

The insoluble is nearly all silica; the ore from the upper levels of the Chispas vein, however, carries varying amounts, up to 3 or 4 percent. of aluminum silicate, as does the ore from one of the parallel veins.

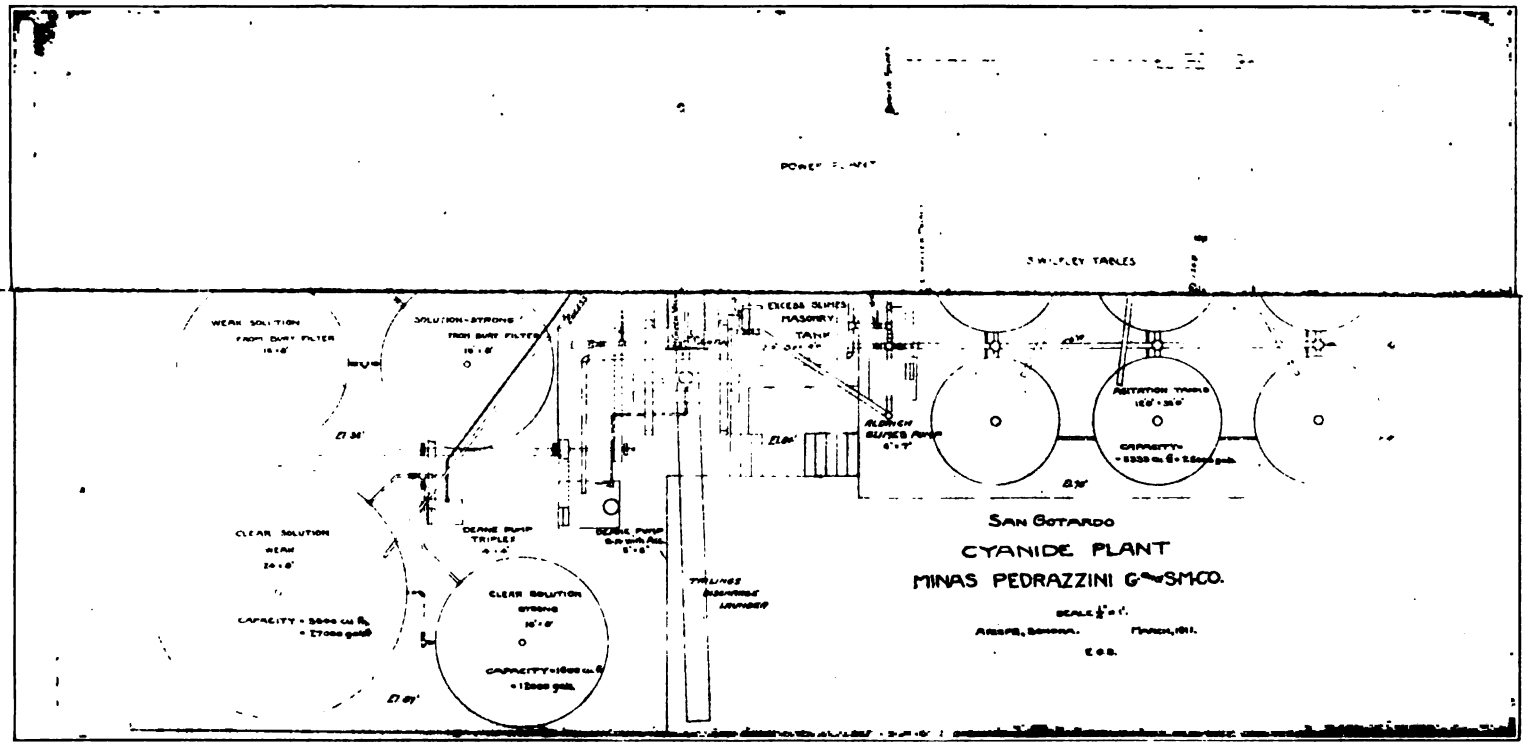
The cyanide experiments with slimes from the accumulation in the ponds below the former mill showed an extraction of 93 percent. of the silver and 98 percent. of the gold, on material which had been long exposed to the weather, and an extraction of slightly over 90 percent. both of gold and silver from the coarser sand tailings which had not been so long exposed. The experiments further showed that the cyanide consumption would probably not exceed 3 lb. per ton of tailings and slimes treated, that about 0.2 lb. of lead acetate would be required per ton, slightly under 2 lb. of lime per ton to neutralize acidity, and that 36 hours' agitation by air would be required to effect the solution of values if the pulp were all slimed to pass at least 150-mesh, to which fineness it was found necessary to grind the tailings in order to reach 90 percent. extraction.

The crushing section of the former mill, consisting of a 7 × 10 Blake crusher, and 20 stamps of 850 lb. dropping 100 times per minute with a 6½-in. drop, crushed 46 short tons per 24 hours through a 20-mesh screen of No. 25 wire (aperture 0.027 in.). As about 26,000 tons had been treated by this mill up to the end of 1910, and of the tailings discharged over 12,000 tons had been

saved in slime ponds, besides several thousand tons of lower-grade sand tailings, it was necessary to design a cyanide plant that would be sufficient to treat the current discharge of tailings from the concentrating tables (having decided to discard the amalgamation process) and at the same time to treat the accumulated slimes and tailings of the former mill. Allowing for ordinary mill stoppages, it was calculated that at least 40 tons per day of current mill tailings would have to be cyanided, and the plant was designed to treat in addition a minimum of 30 tons per day, to be taken from the accumulated slimes and sand tailings.

It will be shown at what estimated capacity each machine in the new plant was taken and what allowances were made for extra work to be thrown upon any part of the plant. Due to the fact that a high extraction by cyanide could be expected, it was thought wise to change the mesh of the battery screen from the 20-mesh above noted, to 16-mesh with No. 22 wire, giving an aperture of 0.033 in., increasing in this way the stamp duty to possibly 3 tons per stamp per 24 hours. The four batteries of the mill have not yet all been equipped with the new screen, to determine whether the expected stamp duty will be attained. It is my opinion that this duty will be exceeded. The coarser feed to the Wilfley tables will of course result in a lower saving by concentration and richer table tailings, but as all the table tailings will be slimed in the tube-mill, the ultimate saving will be greater, since the smelter, in buying the concentrates, pays only for 95 percent. of the silver content. It is also expected to give the tables a greater inclination and make a higher-grade concentrate, so as to effect a saving in overland freight to the railroad. The smelter charges for briquetting and smelting concentrates, together with railroad freight and overland freight to the station, amount to $46 American, per ton, whereas the refining charge on silver-gold bars is only 1 c. per oz. (payment for full gold and silver contents) and the express charges on bullion are $4.25 per thousand ounces.

One of the appended plates gives the ground plan of the cyanide addition to the mill. For purposes of greater clearness, in preparing this plan some of the pipe-lines have been omitted and a number of the launders. The positions of the 20 stamps,


POWER PLANT
3 WILFLEY TABLES
WEAK SOLUTION FROM DIRT FILTER
SOLUTION-STRONG FROM DIRT FILTER
EXCESS SLIMES MASONRY TANK
ALDRICH SLIMES PUMP
AGITATION TANKS
CLEAR SOLUTION WEAK
CLEAR SOLUTION STRONG
DENNE PUMP TRIPLEX
DENNE PUMP
TAILINGS
SAN GOTARDO
CYANIDE PLANT
MINAS PEDRAZZINI G. & S. M. CO.
ARIZPE, SONORA. MARCH, 1911.
E. & S.

the concentrating tables and the power plant are only indicated in outline. The cyanide plant consists of the following machines and appliances: One 14-in. belt conveyor, 160 ft. long, running at a belt speed of 105 ft. per min., brings the slimes up from the ponds below the old amalgamation plant to a masonry mixing tank, 16 ft. in diameter. The current mill tailings, after leaving the Wilfley tables, are dewatered and also discharge into this mixing tank. A bucket elevator, with 5 × 8-in. buckets, raises the combined tailings and slimes to a Dorr standard simplex classifier.

Below the Dorr classifier is located an Abbé tube-mill, 4½ × 20 ft. with trunnion bearings, and furnished with El Oro type of lining. To facilitate transportation by carts, this tube-mill was made in two parts, the shell being cut in two and each half having a ring riveted to it, carefully faced, so as to give exact alignment when the two rings were bolted together on the ground. Danish pebbles are used in the tube-mill, the initial charge being 16,000 lb., half of which were of the No. 3, or largest size, the remainder being about 5,000 lb. of No. 2 and 3,000 lb. of No. 1 size; subsequent feed, of course, is all of No. 3 size. Such additional water as is needed in the tube-mill, for maintaining the proper consistency for grinding, is added as weak solution; the stamp milling is done in water in order to avoid consumption of cyanide by the metals removed with the concentrates.

At the discharge end of the tube-mill is a second Dorr classifier. Above the tube-mill building are located two stock solution tanks of steel, 24 ft. diameter and 8 ft. deep, for strong and weak solutions, capacity 27,000 gal. each. Outside of the tube-mill house, and at a slightly lower level, is situated a Dorr thickener in a steel tank 24 ft. in diameter and 8 ft. deep. Beyond this, as there was not enough fall available, it was necessary to put in a short air-lift which is easily operated, on the thickest slimes handled, by about 4 lb. of air pressure, which raises the thickened slimes to the box from which they are distributed to the six agitation tanks. These tanks are of steel 12 ft. in diameter and 36 ft. high, with a conical bottom sloping 60°, having a capacity of 25,000 gal. each. The detail of the construction of these 36-ft. tanks is as follows: The top ring is composed of No. 10 steel, four pieces

to the circle, 8 ft. high, single riveted at the top to a 3 × 3-in. angle iron, single riveted at the lower side to the second ring, which is 6 ft. 6½ in. high and is made of No. 9 steel. Below this follows the third ring 6½ ft. high, made of No. 8 steel. At the bottom of this third ring is a double row of rivets fastening the third ring to the fourth, the fourth ring being of No. 7 steel, six pieces to the ring. This fourth ring is 6 ft. 8 in. high and is fastened by a double row of rivets at the bottom end to a 6 × 4-in. angle iron. The cone joins at this point, being riveted both to the sheet of the fourth ring and to the angle iron. The cone is made of ¼-in. steel in six pieces. The height of the cone is 9 ft. 8 in. and at the bottom end is riveted by a double row of rivets to a 10-in standard cast-iron Tee, on one side of which is a blind flange and on the other side a 10-in. gate valve. The bottom of the Tee is supported on concerte. There is no mechanical agitating device in these agitation tanks, but they are provided with the Towne patent air agitator. This consists of a central pipe 10 in. diameter, reaching nearly to the bottom of the cone and perforated by rows of 4 holes, 1½ in. diameter, the rows of holes being 1 ft. apart except at the upper end of the pipe, where, for 2 ft., they are 6 in. apart. Almost at the top level of the tank there are four 2-in. pipes leading outward and spirally curved from the central pipe towards the circumference of the agitation tanks. The central pipe extends 2 ft. above the top of the tank and is open at the top. Air is admitted through a 1½-in. pipe passing through the blind flange in one side of the 10-in. Tee at the bottom of each cone. The air pipe enters the center of the central pipe and is turned upward inside of it for about 18 in., terminating in a vertical ball check-valve. The discharge from each agitation tank is controlled by a gate valve, all tanks connecting into one central pipe leading to the suction of an Aldrich triplex slimes pump, of 6 × 7 in. size.

On one side, and above the Aldrich slimes pump is situated the Burt filter, 42 in. diameter by 20 ft. 8 in. long. The Burt filter is an iron cylinder, inclined 45°, into the lower end of which the slimes are forced under a pressure of 40 to 50 lb. per square inch. In the cylinder are suspended the filter mats of cocoa matting covered with canvas, their frames being perforated pipes con-

Completed Cyanide Plant, May 25th, 1911

nected to a common discharge pipe running along the top of the cylinder out of which the filtered solution passes. The pressure being positive and three times the atmospheric pressure being obtainable, this type of filter is preferable for slimes containing high silica and little alumina, and is quicker in operation than the vacuum types.

A masonry tank, 8 × 8 ft., 4 ft. deep, is provided at one side of the Burt filter to receive the excess slimes discharged from the filter after the cake is formed and before the weak solution is forced through the cake for washing. The strong and weak filtered solutions are collected in the respective tanks beyond the Burt filter, these tanks being of steel, 16 ft. in diameter and 8 ft. deep, capacity 12,000 gal. each. The solutions are drawn from these tanks for precipitation by zinc dust, which is added by means of two types of zinc feeders, a Colbath and a Creston-Colorada, which are placed inside the zinc room. A filter press of 12 leaves, 24 × 24 in., made by the Creston-Colorada Company is used for the product. The barren solutions are run to two sump tanks outside the building, these being of steel and of 24 ft. diameter for the weak solution and 16 ft. for the strong, both being 8 ft. deep. A Deane triplex pump is used for pumping the zinc and solution into the product press, and Deane triplex pumps are also used for returning solutions to the stock tanks. Power for all the triplex pumps and for the feeders is provided by a General Electric Company's type DLC, direct-current, 220-volt, motor of 25 kilowatts, located in the zinc room. All the other power is derived from the main line-shaft of the mill, which is driven by a 150-h.p. Hamilton-Corliss engine. The generator, which supplies the current for the motor as well as for the lights and for the assay office motor, is located in the main power plant and consists of a Buffalo Forge Company's engine of 90 h. p., direct connected to a General Electric Company's form RB-6 pole, 250-volt, generator of 50 kilowatts. Compressed air is furnished by two compressors built by the Chicago Pneumatic Tool Company, and about 20 lb. of air pressure is maintained. The normal amount of air furnished is 320 cu. ft. of free air per minute, which however, will probably prove too small if more than four of the agitation tanks are being operated at one time; air is also

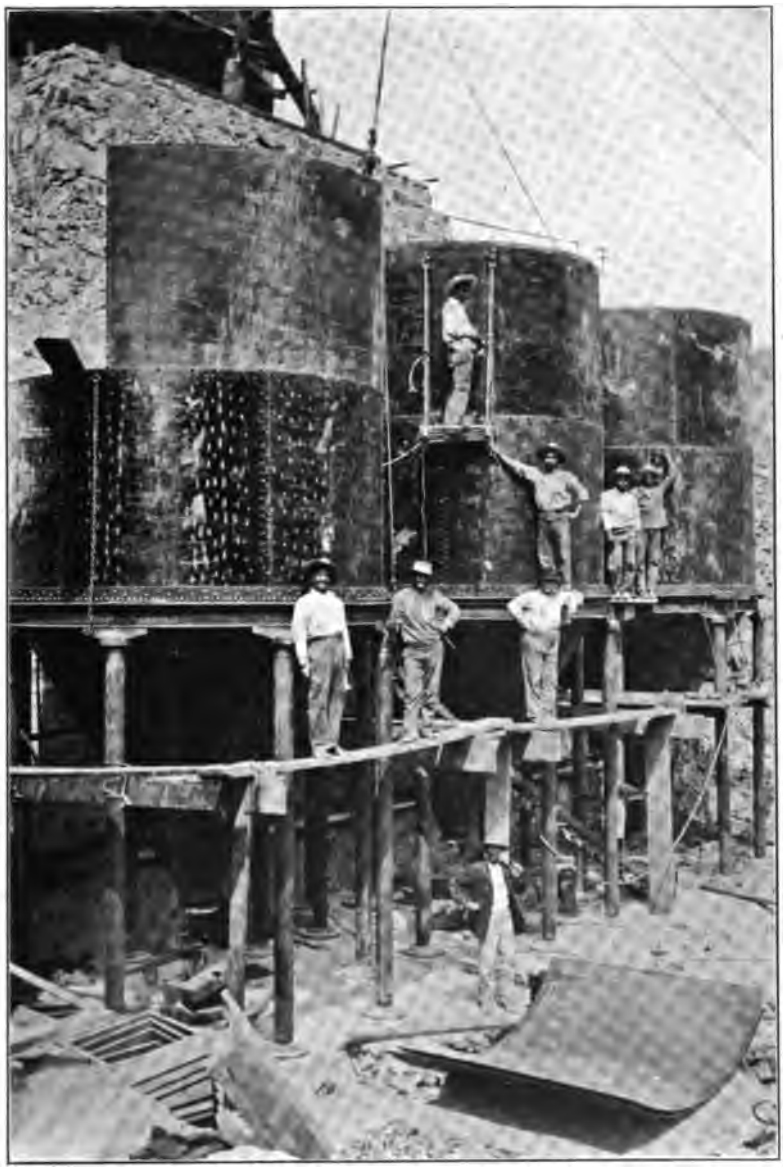
needed for the Burt filter and for the air-lift, the lower pressure therein required being obtained through a reducing pressure valve. It is to be noted that all the valves, cocks and glands or stuffing boxes that are to be in contact with cyanide solution are of iron, no brass being permissible. The masonry tanks are faced with cement, which was then washed with ammonium carbonate solution and then painted with cyanide paint.

In addition to the appliances shown in the plan, there are three tanks, one a steel tank 6 × 8 ft., located on the mountain side, 290 ft. above the Burt filter, for accumulating hydraulic pressure to operate the filter door, and the others are masonry tanks located at about 80 ft. above the mill, for excess solution and for slimes which also are required to be under pressure, as will be shown in describing the operation of the plant.

Four of the other plates reproduced with this paper show the various stages in the construction of the plant, from the start of the foundations to the completion of the mill. The other two plates show the detail of the construction, and method of support, of the agitation tanks and the general arrangement of the Burt filter.

Due to the situation of the camp, with a 72-mile wagon haul from the railroad at Cananea, all the tanks were shipped knocked-down and the riveting was done on the ground. Ingersoll riveting hammers were used, the compressor for the air agitation furnishing the compressed air. It was somewhat of a risk to use this compressor, which was designed for a maximum of about 50 lb., but had to be run at 80 lb. to furnish effective air pressure for the riveting hammers. The stock and solution tanks are of 3-16-in. steel, both sides and bottom, and are painted with cyanide paint, two coats inside and one coat outside. They rest on rough rubble masonry foundations, timber being out of the question, as the wagon freight alone on rough pine amounted to 90 pesos per thousand feet.

The ordinary Pachuca type of agitation tank, so far as I had seen it installed in other places, seemed to involve a waste of material in having the cylindrical sides carried to the ground, with the cone at the bottom built inside; such construction also leaves the bottom of the cone, with its valves and connections,



Construction of Agitating Tanks, March 24th, 1911

rather inaccessible. I therefore decided to support the agitation tanks at the top of the cone, by using a 4 × 6-in. angle iron, which is riveted to the tank sheets. This angle iron rests on a curved I-beam, 7-in. standard section, made in four pieces, each one-quarter of the circumference of the outside of the tank. This circular I-beam is carried on eight columns, made of 6-in. pipes, with special castings for bases, and capped by castings terminating in a flange conforming to the circle of the I-beam. The pipes are filled with cement concrete and the castings are threaded on, but care was taken to have the ends of the pipe "bottom" into the castings so that the strength of the column would not rely on the strength of the thread. These pipe columns are 12 ft. high and are braced laterally at half their height.

The agitation tanks are set up in three rows of two, and are held together at the top of the cones and at the top of the tanks by steel plates fastened to the angle-irons which encircle the tanks. The whole set of tanks is anchored by six horizontal bolts to the retaining wall which sustains the ground forming the bench carrying the Dorr thickener. There are no special features about the Dorr thickener and the Dorr classifiers, which are the standard machines furnished by the manufacturers, but for the benefit of those not familiar with these appliances, it may be stated that the classifier is a steel box with an inclined bottom up which a set of reciprocating hoes draws the settled sands towards the discharge end; before reaching the discharge the sands pass over a series of jets of water which clear them of the last traces of slimes. The deeper end of the classifier box is provided with a baffle board under which the flow of water and fine slime passes to the overflow compartment, or discharge of the undersize. The hoes move at the rate of 12 strokes per minute, the length of stroke being adjustable. One Dorr simplex classifier at the head of the tube-mill was thought to be sufficient to handle the entire feed that will go to the cyanide plant, on the assumption of 30 tons of the old slimes per day with 40 tons of the current mill table tailings of which over 40 percent. will be slimes, but it is probable that the sizing capacity will not be found sufficient when the 20 stamps are all operating on the new 16-mesh screen, and in order to take out all the material of the size to be fed to the

tube-mill it will probably be necessary to install an additional classifier. On the other hand, the Dorr thickener at the discharge end of the tube-mill, which will take out the material which has not been ground to sufficient fineness, will have a very light load except when the tube-mill is crowded by a large proportion of coarse sands coming from the old slime ponds. The Dorr thickener is simply a tank of large diameter into which the slimes are fed near the center and just below the surface of the water, all the solids supposedly being finer than 150-mesh, and probably 80 percent. of them are 200-mesh. It was considered that 24 ft. diameter, with 8 ft. depth, was sufficient to allow all of the 70 tons of solids per 24 hours to settle in a thickened slime to the bottom, and a clear solution to overflow the circular launder around the edge, without causing currents to interfere with the clarifying. The discharge of slimes is continuous and issues from the center, towards which a series of rakes, set diagonally, slowly moves the slimes across the bottom of the tank. The arms carrying the rakes make one revolution in eight minutes. The thickening process is necessary, as the undersize from the first classifier contains a large amount of water, and the discharge from the tube-mill and the second classifier contains too much weak solution to allow the pulp to be passed in this condition to the agitation tanks, since we there require a pulp of 1:1½ or 1:2, with strong solution.

The capacity of the agitation tanks was figured on a basis of about 35 tons of dry slimes to each tank, two tanks being filled in each 24 hours; allowing 36 hours for agitation and 12 hours for settling and decantation, we would always have one pair of tanks emptying and filling, with two pairs of tanks agitating on any given day, thus discharging 70 tons of slimes as a minimum to the Burt filter in each 24 hours. It is found that the upper 8 ft. of the contents of a tank will settle in a few hours sufficiently clear to allow decantation, the clear solution not requiring to go to the Burt filter; for that purpose a 4-in. decanting pipe is provided 8 ft. below the top of each agitation tank. The only clarifying that this solution needs before going to the zinc precipitation is provided by sand filters on the false bottoms of the two 16-ft. tanks where the pregnant solutions are held.

Cyanide Plant Construction, March 24th, 1911

The capacity of the Burt filter of the size adopted (42 in. × 20 ft. 8 in.) was more problematical, as it depends entirely upon the rapidity with which each operation on the cycle of filtering is conducted. The Aldrich slimes pump, if driven so rapidly as to effect a quick filling of the filter, also produces an excessive pressure, and it was decided, at the suggestion of Edwin Burt, to put in one of the masonry tanks on the hill at 80 ft. elevation, which is connected with the discharge of the slimes pump so that the pump could run continuously and yet during the time that the filter was filling the excess slimes collected in the upper tank would run back quickly to the filter. The original calculation for the Aldrich pump (6 × 7-in. triplex) had been 100 gal. per minute; taking an agitation tank of 25,000 gal., from which about 6,000 gal. had been decanted, there would remain 19,000 gal. which could be taken care of by the Aldrich pump in about 3½ hours, allowing the balance of the time out of each 12 hours for forming the cake, pumping through the wash solution and discharging the washed filter cake. The Burt filter, operated under the original condition, was filled, washed and discharged once in about 50 minutes, but it is hoped to improve this with the use of the high tank for quicker filling with slimes, and in this connection the second of the high masonry tanks (80 ft. elevation) is used for weak solution to effect a quick washing of the filter cake (four minutes). The filter should then easily handle over 70 tons, possibly 80 tons, in 24 hours. Increased capacity is also expected by the use of thicker pulp in the agitation tanks; at the present writing, charges of about 38 tons of dry slimes per tank are being experimented with, in place of 35 tons.

Experimental work is still going on to determine exactly the best method of mixing the zinc dust with the solutions to be precipitated. Two types of feeder are being used, as stated in the outline of the machinery adopted. Variations in the manner of applying the zinc to the solution feed pipes still have to be tried, so that a more detailed description had better be left for a later paper, in which I hope to give exact costs of operation.

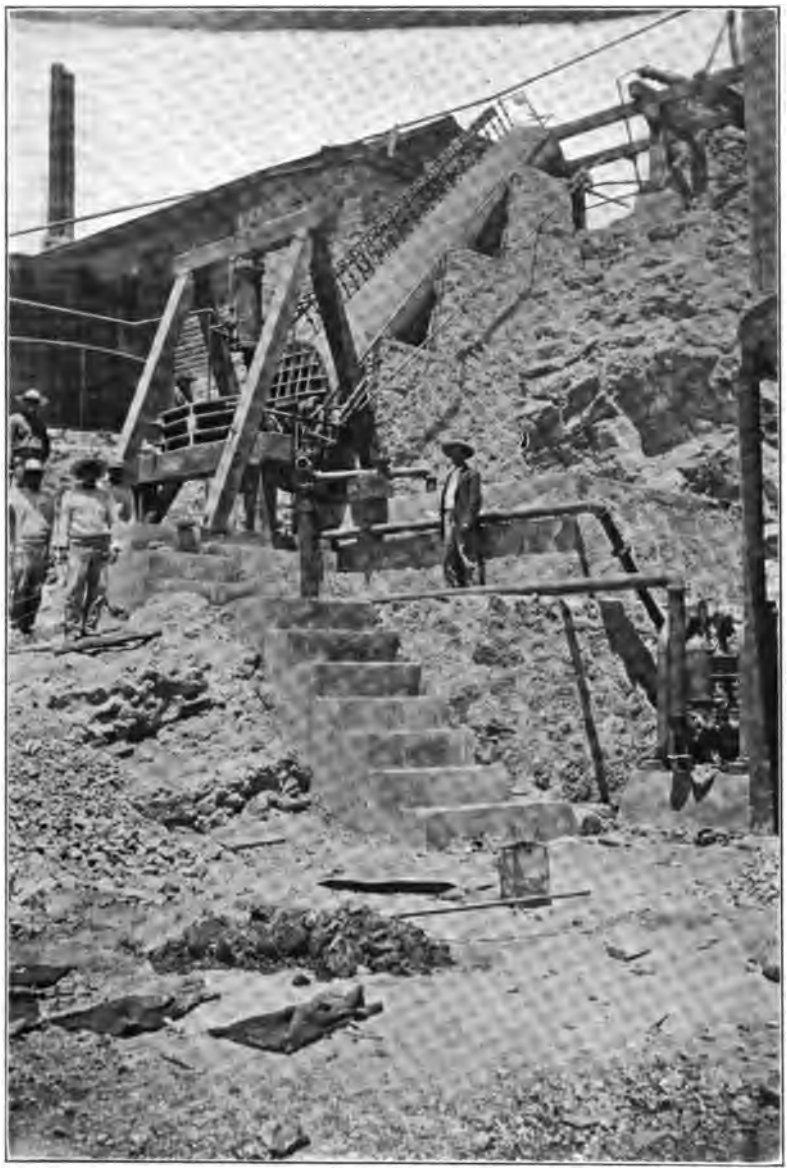
The filtered precipitate is dried in pans in a reverberatory hearth, heated by a very light fire. After drying, and mixing

with fluxes, the melting and refining is done in No. 60 graphite crucibles in two coke furnaces.

The entire plant was constructed without interrupting the operation of the concentrating mill and the old amalgamation process. The construction of the cyanide plant necessitated the building of a new assay and chemical laboratory, to make room for the new concentrate house, this being required as the concentrate was formerly dried, sampled and sacked in the house occupying the space taken for the tube-mill. This gave an opportunity for improving the method of handling concentrate. Formerly this was shoveled from boxes at the end of the Wilfley tables and removed in wheel barrows, whereas at present the concentrate drops through a launder and pipe to a lower level and is dewatered in masonry tanks and then shoveled to a steel drying plate 8×24 ft., which is kept hot by the exhaust steam from the compressor.

The time consumed in erecting the plant was very much prolonged by the outbreak of the Mexican revolution. Labor conditions were very much upset, many of the blacksmiths, riveters, mechanic's helpers and masons leaving their work to join the revolution; the efficiency of those remaining, or of the new men secured, was only about half of the normal. Many shipments of machinery were delayed through suspension of train service at the border, and through the temporary closing of the Mexican Custom Houses. After the machinery reached the end of the railroad, there was great difficulty in securing a sufficient number of freighters to haul to the mines, since at that time the revolution was at its height and raiding parties were daily seizing all available horses and mules. Notwithstanding the delays thus occasioned, the plant was completed in just under six months from the time the excavation was ready for the masons to start. A number of changes in pipe lines, launders and tanks were subsequently made and more will undoubtedly be necessary as experience is gained with the particular type of ore to be treated. At present, with only a short experience yet available, I would make the following criticism of the plant as we have erected it:

1. Probably the classification at the feed end of the tube-mill is of insufficient capacity.



Burt Filter Plant

2. Probably the compressed air supply is too small.

3. Additional capacity to the Burt filter plant may be found necessary.

4. Probably a large product filter press in the zinc room would be found advisable.

These imply changes to the plant which are not necessarily required at once, nor does their lack interfere with successful operation. The ratio of the capacity of each machine to that of the entire plant may vary with conditions that will arise from time to time. For instance, if any of the four batteries of the stamp mill be hung up for repair, the feed can at once be increased on the conveyor bringing up the slimes. The conveyor provided for this purpose was of a much larger capacity than needed for elevating only the proportion of old slimes which it was aimed to treat under normal conditions. The fact that a large part of the old accumulation of coarse sand tailings can be cyanided at a profit, will result in their being brought up with the slimes as circumstances permit, and this too will vary the conditions under which the classifiers will work. The tube-mill is of larger capacity than normal running conditions would require, but it was thought well to install it at once of a size that would grind all the mill tailings, if after the accumulated slimes shall all have been cyanided, it should be decided to increase the stamping and concentrating section of the mill to a size to supply tailings up to the capacity of the cyanide plant.

[Contribution from the Havemeyer Laboratories of Columbia University, No. 200].

THE RELATION OF CHEMICAL COMPOSITION TO CALORIFIC POWER IN WOOD, PEAT, AND SIMILAR SUBSTANCES.

BY H. C. SHERMAN and C. G. AMEND.

The publication in a recent QUARTERLY [1] of Mr. Myers' investigations of tan bark as a boiler fuel adds interest to the question of the relation of chemical composition to calorific power in fuels of this general nature.

Myers reports on the dry matter of average, spent, hemlock tan: Ash, 1.42; hydrogen, 6.04; carbon, 51.80; oxygen, 40.74 percent.; calorific power, 9504 B. t. u. per pound. By the use of properly designed furnaces he effected considerable improvement in thermal efficiency, up to 71.1 percent. Myers' calculations of efficiency are based on actual determinations of calorific power. If an attempt be made to calculate the calorific power from the composition by means of the usual formula:

$$x = 8080C + 34500\,(H - \tfrac{1}{8}\,O) + 2220S$$

we obtain, $x = 4513$ calories, or 8123 B. t. u.

It is evident that an attempt to compute calorific power by the usual formula instead of by direct determination would in this case have given values some 14.5 percent. too low, with the result that the estimated thermal efficiency would have been correspondingly too high.

A knowledge of the real relation between chemical composition and calorific power would not only be useful in future calculations, but would make possible the recalculation of results of earlier experiments, in which the chemical composition of the fuel may have been correctly determined but its calorific power erroneously estimated, so as to permit of their comparison with the more recent and accurate data, such as those of Myers.

The formula of Dulong above given, and the various modifica-

[1] SCHOOL OF MINES QUARTERLY, Vol. 31, p. 116.

tions which are to be found in the different handbooks, are based on "Welter's rule" which assumes that the oxygen present in the dry fuel is combined with the hydrogen, or that we shall offset the heat of formation of the oxygen-containing molecules if we deduct the quantity of heat which would be produced if the oxygen present combined with hydrogen.

Walker,[2] in 1899, stated that "a better result may usually be obtained by subtracting the oxygen, not with the corresponding quantity of hydrogen, but with the corresponding quantity of carbon, and then estimating the heat of combustion of the elements in the residue."

Miller[3] referred to this suggestion, attributing it to Ostwald, but so far as the writers are aware it has never been applied practically.

It seems therefore worth while to place on record the data obtained in a brief study of this relationship in some woody fuels analyzed in this laboratory.

The results of ultimate chemical analysis, reduced to the basis of dry matter, are given in Table I.

TABLE I

ULTIMATE CHEMICAL COMPOSITION OF DRY MATTER

Substance	Carbon %	Hydrogen %	Oxygen %	Nitrogen %	Sulphur %	Ash %
Chestnut wood chips	50.28	5.58	43.21	0.10	0.03	0.80
Chestnut wood chips, leached	50.09	5.65	43.33	0.10	0.02	0.81
Hemlock tan	53.74	5.66	39.05	0 24	0.04	1.27
Hemlock tan, leached.....	54.97	5.73	37.69	0.26	0 02	1.33
Oak tan, leached	49.51	5.53	39.24	0.40	0.05	5.27
Bagasse	49.04	5.96	42 58	0.31	0.07	2.04
Oil Cake	48.20	6.47	38.02	4.24	0.25	2.82
Peat	57.30	4.66	19.26	1.13	0.77	16.88

From these analyses the calorific power, or heat of combustion, was estimated (1) according to Dulong's formula given above,

[2] Introduction to Physical Chemistry, p. 123.
[3] Calculations of Analytical Chemistry, 3rd Ed., p. 135.

which is based on Welter's rule of calculating the oxygen with the hydrogen, (2) according to Walker's suggestion of calculating the oxygen with the carbon. Using here the same calorific values for the elements as in Dulong's formula we have:

$$x = 8080(C-\tfrac{3}{8}O) + 34500H + 2220S.$$

Table II shows the results (in calories) obtained by these two formulas, along with those determined directly by means of the Atwater-Mahler bomb calorimeter.

TABLE II

ESTIMATED AND DETERMINED CALORIFIC POWERS

Substance.	Welter's rule	Walker's rule	Direct determination.
Chestnut wood chips	4125	4659	4632
Chestnut wood chips, leached	4129	4684	4661
Hemolck tan	4612	5113	5106
Hemlock tan, leached	4794	5277	5217
Oak tan, leached	4217	4720	4798
Bagasse	4185	4731	4625
Oil cake	4594	5078	4953
Peat	5422	5671	5606
Average	4510	4992	4950

It will be seen that in the case of these woody fuels high in oxygen the calorific values estimated by means of Dulong's formula based on Welter's rule are all much too low, the estimated values in terms of the actual ranging from 88.4 to 96.7 percent. and averaging 90.4 percent.; while the values estimated by a corresponding formula based on Walker's rule show a fair approximation to the values actually determined by the bomb calorimeter, the estimated values in terms of the determined ranging from 100.1 to 102.5 percent. and averaging 100.8 percent.

The two methods of calculation were also applied to seven analyses of lignites taken from *Bulletin 290* of the U. S. Geological Survey. In one of the seven cases the calorific power actually determined was lower than that found by Welter's rule; in one it was materially higher than that found by Walker's rule; in each of the other five cases and also in the average of all seven the results by Welter's rule were much too low, while those cal-

culated according to the suggestion of Walker were approximately correct.

It may be concluded:

(1) That too much reliance should not be placed upon estimates of calorific power from ultimate chemical composition in fuels high in oxygen;

(2) That Dulong's formula, or any similar formula based on "Welter's rule" of calculating the oxygen with the hydrogen, is likely to give results much below the truth;

(3) That the higher results obtained by calculating the oxygen of the sample as combining with the carbon, according to the suggestion of Walker, are much more nearly correct, and in most cases show a fair approximation to the values directly determined.

Department of Chemistry, Columbia University.

October 10, 1911.

VOLATILIZATION OF GOLD AND SILVER FROM THEIR COPPER ALLOYS.*

BY EDWARD F. KERN † AND ALBERT A. HEIMROD.††

INTRODUCTION.

Metallurgists have suspected that a loss of gold occurs as the result of volatilization during the refining of auriferous copper bullion, especially during the operation of poling, when the molten metal is in a state of violent ebullition. It is only within recent years, however, that this hypothesis has received serious attention. The refiners, who maintained that a loss of gold occurred during the refining of copper bullion in a reverberatory furnace, were formerly criticised because they could not present ample proof, the only available evidence being the scoria, the flue dust, and the walls and hearth of the furnace, all of which can be treated for their metal contents. They, therefore, began to hunt for positive proof, and were rewarded when they discovered a brick-red, microscopic deposit, which is washed by rain from the roofs of refinery buildings after a long dry spell. Some of this material was collected and was found to contain a high percentage of copper, but the amount that could be collected was too small to be assayed for gold and silver.

Within the past year, the superintendent of one of the large copper refineries near New York has compiled data for a month on a 130-ton, copper anode furnace. During this period, 25 charges of blister copper (6,500,000 lb.) were melted, fined, poled, and cast. The flue-dust collected during this time weighed approximately 1000 lb. The blister copper assayed between 200 and 250 oz. of silver and 1 to 3 oz. of gold per ton, and also contained from 0.5 to 0.8 percent. of lead. The flue-dust contained 100 oz. of silver, and 0.7 oz. of gold per ton. The loss of metal by volatilization was not a serious matter, as the dust was collected and treated. The volatilization of the gold and silver, in

* This article is based upon an investigation conducted in the metallurgical laboratory of the School of Mines, Columbia University, New York City.
† Assistant Professor of Metallurgy, Columbia University.
†† M. A. Columbia, 1910.

this case, was attributed to the lead in the bullion, which was oxidized and volatilized during the scorification, or fining, period. No data were available as to the fume which passed from the chimney into the atmosphere.

Professor Arthur L. Walker, referring to a case which occurred at a plant of which he was in charge, when copper matte rich in gold was brought up to blister copper by fusion in open reverberatory furnaces, states that gold was undoubtedly lost in these operations by passing off in the gases during the long period required for smelting. This was proved by the fact that condensed fume taken from the top of the furnace stacks contained more gold in proportion to its copper content than was present in the original copper matte. In the final cleanup after treating this matte, it was ascertained by totalling the products and by-products that a certain amount of gold, which the matte originally contained, could not be accounted for, unknown losses having occurred during the treatment. The matte contained 10 oz. of gold to the ton, which, when smelted to blister, produced bullion containing 25 oz. of gold to the ton of copper.

It was at the suggestion of Professor Walker that this investigation was undertaken, the object being to determine whether gold and silver are volatilized when alloys of these metals with copper are subjected to temperatures above their melting point, in a reducing atmosphere.

Abstract of Literature.

J. Napier[1] in 1857 found that gold is volatilized when an alloy of gold with three times its weight of silver is cupelled with lead. He took 20 to 100 grams of gold, subjected the alloy to cupellation for three to eight hours in a muffle furnace, and found varying losses. He also heated alloys of gold and copper in fire-clay pots, and found that these metals volatilized. At the end of the heating, the pots were removed and the remaining alloys assayed by adding 2.5 times their weight of pure silver, cupelling with lead and parting with nitric acid. The results of these latter experiments are given in the subjoined table.

[1] *Journal* American Chemical Society, X, 1857, 229.

The results show, first, that the more copper present in the alloy the greater the loss of gold by volatilization; and secondly, the higher the temperature the greater the losses.

Alloy No.	Wt. of gold in grams.	Wt. of copper in grams.	Time of fusion in hours	Loss of gold in grams.	Approx. temperature.
1	100	12	6	0.234	Only sufficient to melt alloy.
2	100	10	4	0.210	A little higher
3	100	20	4	0.450	than No. 1.
4	100	10	6	0.800	As high as could be obtained in muffle.

According to T. K. Rose,[2] gold is rapidly volatilized when heated before a strong blast of an ordinary blow-pipe, as was proved by Robert Hare. Rose confirmed these results by heating 0.5 gram of pure gold, finding that one-half of it was lost in one hour when heated in a cupel by the flame of a blow-pipe, the temperature being probably less than 1300° C. The conclusions drawn by Rose from experiments conducted at the Royal Mint in London are: 1, loss of gold increases with temperature, being four times as much as 1250° C. as at 1100° C., while it is insignificant at 1075° C., and *nil* at 1045° C.; 2, an atmosphere of carbon dioxide favors the volatilization of gold, the rate of loss being six times as great as in an atmosphere of coal-gas at the same temperature; 3, the substitution of a graphite crucible for a bone-ash cupel increases the loss, whereas clay crucibles have the opposite effect; the increased volatility of gold when alloyed with copper was confirmed, the higher the percentage of copper in the alloy the greater the loss of gold.

Kraft and L. Bergfeld[3] conducted experiments on the volatilization of metals in a vacuum, and found the lowest volatilization point of copper to be 960° C., and of gold, 1070° C.

Moissan[4] subjected metals to high temperatures in an electric-arc furnace. He found that copper distilled to the extent of 50 grams in 5 min. when the furnace was operated with an alternat-

[2] "The Metallurgy of Gold," Ed. iv. pp. 5 and 6.
[3] *Deutsches Chem. Central Blatt*, 1905, I. p. 494; also *Bericht Deutsche Chem. Ges.*, Vol. 38, p. 254
[4] *Deutsches Chem. Central Blatt*, 1906, I. p. 125; also *Comptes Rendus de l'Academie des Sciences*, Vol. 141, p. 853.

ing current of 300 amperes at 110 volts. He collected a part of this copper on a copper tube cooled by cold water. The copper condensed as fine filiments and had the appearance of thread-like needles. The metal remaining in the crucible under the electrodes absorbed lime, aluminum, iron and, what is more important, graphite, presumedly from the electrodes. After the current had been stopped, upon cooling, a scum formed on the surface of the metal remaining in the crucible. At the moment of solidification, small crystals of graphite separated, the mass became turbid, and an abundance of gases were discharged. When the mass of copper had cooled it was cut in half, polished and examined under the microscope. Small spherical cavities were seen, which contained brilliant black crystals of graphite.

Moissan[5] also found that gold distilled with ease in an electric-arc furnace, the boiling point of the metal being 2430° C. With alloys of copper and gold he found that copper distilled before the gold. He also found that gold absorbed carbon, which crystallized out again upon cooling. For the first experiment he used an alternating current of 500 amperes at 110 volts, heating 150 grams of gold for 5½ min. and distilled 10 grams. The gold which remained in the crucible disengaged gaseous bubbles during cooling and on solidification. The metallic mass contained cavities, the surfaces of which were coated with a black network of graphite. He also heated alloys of gold and copper, ranging from 10 gold and 90 copper to 50 gold and 50 copper. He found that the loss of copper in each case was greater than that of gold. The results of Moissan's experiments on the volatilization of metals in an electric-arc furnace are given by Oliver P. Watts in the *Transactions* of the American Electrochemical Society, 1907, XII, pp. 141 to 154.

In summarizing the above abstracts, the conclusions are:

First, that losses of gold occur when copper-gold alloys are subjected to high temperatures, the losses being directly proportional to the length of time that the alloys are under heat, and also varying as the excess of temperature to which the alloys are subjected above their melting point.

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[5] *Deutsches Chem. Central Blatt*, 1906, I. p. 328; also *Comptes Rendus de l'Academie des Sciences*, Vol. 141, p. 977.

Second, alloys of copper and gold absorb carbon while in the molten condition, which carbon is partially liberated as gas during the cooling of the alloy and partially crystalizes as graphite during the solidification of the metal.

Third, the loss of gold depends upon the atmosphere in which the alloy is heated. In an atmosphere of carbon dioxide the loss is greater than in coal-gas.

Fourth, the loss of gold is greater when the alloy is heated in graphite crucibles than when heated in bone-ash cupels; when heated in clay crucibles the loss is least.

Preparation of Standards and Methods of Assay.

The two methods ordinarily used for the determination of gold and silver in copper bullion are direct fire assaying, and a combination wet and fire method. As the chances for error in the determination of gold and silver by direct fire assaying are greater, owing to volatilization, and because the proper temperature of scorification is difficult to control, for this investigation the combination wet and fire method was favored for the determination of gold, and an entire wet method for that of silver.

Assay of Gold-Copper Solutions.—One of the main sources of error in the determination of gold in copper bullion (or an alloy) by the combination wet and fire method is the loss of some of the finer particles of gold when the bulky copper solutions are filtered. One means of preventing this loss is to add some salt that will produce a heavy precipitate, which while precipitating will collect all of the suspended fine particles. The salts generally used for this purpose are lead acetate, which added to a solution containing sulphates yields a precipitate of lead sulphate, or silver nitrate, which added to a solution containing a chloride yields silver chloride. Whether the one collecting agent possesses any advantages over the other was not known, wherefore the first series of experiments had the object of finding out which salt is the more suitable.

A solution of gold and copper was made up, each cubic centimeter of which contained 0.17654 gram of copper and 0.000205 gram of gold. Samples of 50 c.c. each of this standard solution were taken and assayed by a modification of the conventional

methods. The general procedure, after the sample had been measured out, was to add 10 c.c. of concentrated sulphuric acid and evaporate to dryness. The mass was then taken up with distilled water and diluted to 250 c.c. Ten grams of ferrous sulphate crystals ($FeSO_4.7H_2O$) was dissolved in distilled water, and added, which precipitated the gold in solution as finely divided metallic particles. To collect these particles, 10 to 20 c.c. of a saturated solution of lead acetate was then added, giving a heavy precipitate of lead sulphate. The solution was filtered and the precipitate washed, dried, and scorified with 20 to 40 grams of test lead. The button was then cupelled and the gold bead weighed. Results of these assays are given in Table I.

TABLE I

ASSAYS OF STANDARD COPPER-GOLD SOLUTIONS

Assay No.	C.C. of Lead Acetate Solution added.	C.C. of conc. HCl added.	Grams of Ag NO_3 in solution added.	Grams of Test Lead for Scorification.	Theoretical amount of Gold. Gram,	Weight of Gold Obtained. Gram.	Difference in Weight. Gram.	Percentage Error.
1	20			20	0.01026	0.00816	0.00210	—20.4
2	20			20	0.01026	0.00982	0.00044	— 4.28
3	10			25	0.01026	0.01026	0.00000	0.0
4	10			25	0.01026	0.01004	0.00022	— 2.14
5	10			25	0.01026	0.01024	0.00002	— 0.19
6		5	0.5	25	0.01026	0.01010	0.00016	— 1.50
7		5	0.5	25	0.01026	0.01026	0.00000	0.0
8		5	10.0	40	0.01026	0.01017	0.00009	— 0.87
9		5	10.0	40	0.01026	0.01015	0.00011	— 1.08
10		5	10.0	40	0.01026	0.01015	0.00011	— 1.08
11	10			40	0.01026	0.01013	0.00013	— 1.26
12	10			40	0.01026	0.01000	0.00026	— 2.53
13	10			40	0.00718	0.00700	0.00018	— 2.50
14	10			40	0.01026	0.10200	0.00006	— 0.56
15	10			40	0.01026	0.01016	0.00010	— 0.97

More satisfactory results were obtained when a silver nitrate solution was used instead of lead acetate, the silver being precipitated as chloride. The beads obtained by scorification and cupellation were then parted with dilute nitric acid and the gold annealed before weighing.

Although the results were fair, the errors were not constant,

the greater variations occurring when lead acetate was used, and the smaller when silver chloride was the collecting agent. The working out of a standard method for the determination of gold in copper was next undertaken.

Preparation of Standard Copper-Gold Alloys.—The standard alloys were prepared by melting 1500 to 2000 grams of electrolytically refined copper in a graphite crucible, the copper being completely covered with charcoal. A calculated amount of gold was added to five or ten times its weight of molten copper in a small graphite crucible. After maintaining the alloy in a molten condition for a few minutes, it was poured into the large crucible containing the major portion of the copper. The larger bulk of copper was first thoroughly poled to tough pitch and then the rich alloy added. The alloy was well stirred with a stick of graphite, and then poured from an elevation of 6 to 8 ft. into a tub of cold water, having a depth of 12 in. It was found that by granulating the alloy segregation of the gold (and of silver) was overcome. These granules, varying from less than 1 mm. to 1 cm. in diameter, were the standard alloys used in working out a method of assaying, and also for the heating experiments.

Assay of Copper-Gold Alloys.—An alloy containing approximately 100 oz. of gold to the ton of copper (alloy No. 1) was used for the first series of assays. An assay ton (29.166 grams) was weighed out, and dissolved in 90 c.c. of concentrated nitric acid. The residue of gold particles was then dissolved by adding 2 c.c. of a mixture of four parts nitric acid and one part hydrochloric acid (aqua regia). When all the metal was in solution, 37 c.c. of concentrated sulphuric acid was added and the solution carefully evaporated to dryness. The mass of crystals was dissolved in distilled water, and the solution diluted to 300 c.c., then 10 grams of ferrous sulphate crystals ($FeSO_4.7H_2O$) dissolved in water was added to precipitate the gold.

In the third assay of Table II, the fine particles of metallic gold were collected in filterable form by adding 10 c.c. of a saturated solution of lead acetate and 0.5 gram of silver nitrate dissolved in water. This gave a mixed precipitate of lead sulphate and silver chloride. In the other experiments the gold particles were collected by adding 0.5 gram of silver nitrate dis-

solved in water and precipitating the silver with a solution of hydrochloric acid. In all cases the solutions were filtered, the precipitates washed, dried, and together with the filter paper were scorified with 45 grams of test lead. The resulting buttons were cupelled and the beads parted with dilute nitric acid. Results are tabulated in Table II.

TABLE II

ASSAY OF STANDARD COPPER-GOLD ALLOYS

Assay No.	Alloy No.	Wt. of Sample in grams.	Grams of $FeSO_4.7H_2O$ added.	C.C. of HCl added.	Gram of $AgNO_3$ added.	Grams of Test Lead for Scorification.	Weight of Gold in gram.	Ounces of Gold per ton.
1	1	29.1667	10	5	0.5	45	0.09577	95.77
2	1	29.1709	10	5	0.5	45	0.09482	94.80
3	1	29.1696	10	5	0.5	45	0.09758	97.57
4	1	29.1660	10	5	0.5	45	0.09622	96.22
5	1	29.1660	10	5	0.5	45	0.08932	89.32
6	1	29.1712	10	5	0.5	45	0.09780	97.78
7	1	29.1685	10	5	0.5	45	0.09725	97.24

The errors in this series of assays (Nos. 1 to 7) were attributed to the fact that the ferrous sulphate did not thrown down all the gold, as it was partially oxidized by the presence of aqua regia. The reason for adding aqua regia was in order to get the alloy entirely in solution, hoping thus to obtain a complete separation of the gold from the copper. By following this procedure, it was thought that if no copper were present with the gold during cupellation, the errors of assaying would be reduced.

In the next series of assays (Nos. 8 to 13) the gold was not dissolved in aqua-regia, and silver nitrate was used as the collecting agent for the gold particles, it being precipitated with hydrochloric acid. The errors in this series of assays (No. 8 to No. 13) were accounted for by the presence of aqua-regia formed by free hydrochloric acid and traces of nitric acid in the solution.

Alloy No. 1 contained approximately 100 oz. gold per ton, and alloy No. 2 about 10 oz. per ton. Table III shows the tabulated results.

Standard Method of Assay.—The method finally adopted for assaying the copper-gold alloys, before and after subjecting them to heat treatment, was as follows: About 1 assay ton of the alloy was carefully weighed, placed in a 900-c.c. beaker, and 20 c.c. of distilled water added. The beaker was covered with a watch-

TABLE III

ASSAY OF STANDARD COPPER-GOLD ALLOYS

Assay No.	Alloy No.	Wt. of Sample in grams.	Grams of $FeSO_4.7H_2O$ added.	C.C. of conc. HCl added	Gram of $AgNO_3$ added.	Test Lead for Scorification, grams.	Weight of Gold, gram.	Ounces of Gold per ton.
8	1	29.1711	10	5	0.50	45	0.09912	99.10
9	1	29.1708	10	5	0.50	45	0.09929	99.27
10	1	29.1680	10	5	0.50	45	0.08936	89.35
11	2	29.1697	10	5	0.05	45	0.00990	9.90
12	2	29.1723	10	5	0.05	45	0.01008	10.07
13	2	29.1722	10	5	0.05	45	0.01030	10.29

glass and a total of 90 to 100 c.c. of concentrated nitric acid (sp. gr. 1.42) was added in quantities of 20 to 25 c.c. from time to time. After all the alloy was dissolved, 40 c.c. of concentrated sulphuric acid (sp. gr. 1.84) was added in order to transform all the copper nitrate into copper sulphate (0.87 c.c. conc. H_2SO_4 theoretically required for 1 gram copper). A mass of light-blue needle-like crystals was formed by the addition of the sulphuric acid, but was easily broken up by stirring and by subsequent evaporation of the supernatent liquid to white fumes.

The solution was evaporated to dryness and fumes of sulphur trioxide by placing the beaker on a sand-bath, so regulating the heat as to prevent bumping. This left a mass of copper sulphate crystals absolutely dry, with part of the water of crystallization driven off. The beakers and contents were allowed to cool and 500 c.c. of distilled water was added, which readily dissolved the salts on heating. Then 10 grams of ferrous suphate crystals ($FeSO_4.7H_2O$) dissolved in water, was added for the purpose of precipitating any gold in solution and to destroy free nitric acid, if present. Next, 25 c.c. of a saturated solution of sodium chloride and enough silver nitrate to make a bead weighing about

eight times the weight of gold present were added. If a greater amount of silver nitrate were used, the bead obtained by cupelling would be too large to part; that is, the gold would be left as a powder instead of a single porous mass. 1.5 gram $AgNO_3$ is equivalent to 1.0 gram silver.

The silver nitrate was added when the copper sulphate solution was hot, and the solution rapidly stirred to make the silver chloride coagulate and settle, thus collecting all the suspended gold in the precipitate. The solution was filtered and the precipitate washed until all copper salts were removed from the filter paper. The beaker was then wiped clean with a piece of filter paper which was added to the main precipitate.

The silver chloride precipitate, containing the gold, together with the filter paper, and about 20 grams of test lead were placed on a litharge glazed scorifier. The filter paper was slowly burnt off, and about 20 grams more of test lead was added, making a total of 40 grams. A pinch of borax glass was placed on top of the lead, and the scorification made in the usual manner. The resulting lead button was cupelled on a medium hard bone-ash cupel, and the gold-silver bead parted with dilute nitric acid

TABLE IV

ASSAY OF STANDARD COPPER-GOLD ALLOYS

Assay No.	Alloy No.	Weight of alloy in grams.	Grams of $FeSO_4.7H_2O$ added.	Gram of $AgNO_3$ added	Grams of Test Lead for Scorification.	Weight of Gold in Grams.	Ounces of gold per ton.
14	2	29.1652	10	0.08	40	0.01044	10.44
15	2	29.1594	10	0.08	40	0.01040	10.40
16	2	29.1676	10	0.08	40	0.01042	10.42
17	2	29.1630	10	0.08	40	0.01040	10.40
18	3	29.1534	10	0.30	40	0.03568	35.69
19	3	29.1511	10	0.30	40	0.03560	35.62
20	3	29.1538	10	0.30	40	0.03566	35.67
21	4	29.1601	10	0.80	40	0.09585	95.87
22	4	29.1598	10	0.80	40	0.09583	95.85
23	4	29.1627	10	0.80	40	0.09584	95.85

(1 part conc. HNO_3 and 8 parts water) followed by a stronger solution (1 part conc. HNO_3 and 4 parts water). The residue was washed with distilled water, annealed, and weighed.

Glazed scorifiers were used in order to prevent loss of gold by particles of the ignited filter paper's becoming attached to the side of the scorifier which would prevent their collecting in the lead button. During scorification the litharge glaze melted and washed the particles of ash into the lead bath, and any gold which the ash contained was collected by the molten lead. The glazing was done by putting 10 to 15 grams of litharge on the scorifier, placing it in a muffle furnace and heating to fusion of the litharge; the excess of litharge was then poured off and the scorifier allowed to cool for subsequent use.

Results of assays made by the above method are given in Table IV.

Volatilization of Gold.

Method of Heating Alloys.—The heating of the copper-gold alloys, in order to determine the loss of gold as the result of volatilization at temperatures above their melting point, was conducted in the following manner: About 200 grams of the alloy was placed in a small Dixon graphite crucible, 3 in. high and $1\frac{3}{4}$ in. inside diameter covered with crushed charcoal ($\frac{1}{4}$ in. and less), and a graphite cover placed on top. Six of the small crucibles, containing the alloy and charcoal, were placed in a Dixon graphite box, $8\frac{1}{2} \times 6 \times 6\frac{1}{4}$ in. inside dimensions. The small crucibles were then surrounded and covered with crushed charcoal, and the box covered with a close fitting graphite top. The double covering prevented the products of combustion and fumes of the furnace from coming in contact with the alloy, while the charcoal maintained a reducing atmosphere. The box was set on bricks sprinkled with crushed quartz in a gas-fired, air-blast furnace. This furnace was capable of developing a temperature of 1650° C.

One set of alloys was subjected to temperatures which varied between 1360° and 1500° C., attempting to keep a constant temperature of 1400° C. A second set of alloys was subjected to temperatures that varied between 1080° and 1150° C. Pyrometer readings, by means of a standard Wanner optical pyrometer, were

taken every 60 to 90 min. during the higher temperature periods, and every 15 to 30 min. during the lower temperature periods. Records of the temperature readings are tabulated in Tables V and VI.

In the case of the copper-gold alloy subjected to the higher temperature, two sets of alloys were used, one running 10.40 oz. of gold per ton (alloy No. 2) and the other 96.04 oz. of gold per ton (alloy No. 5). Three samples of each alloy were taken, and subjected to the heat treatment, making in all six samples, or three duplicate sets of two different alloys. The heating of all three sets was started at the same time; one set was removed after 8½ hours at a temperature of about 1400° C., and the metal was granulated by pouring into a basin of cold water. The second set was removed after being heated for 20 hours; it was granulated in the same manner as the first set. The third set was removed and granulated after heating for 30 hours. The 20- and the 30-hour heats were not continuous but occupied three and five days respectively, the heating being conducted during the day for periods of 5 to 8½ hours.

The alloys subjected to the lower temperature were heated in a similar manner. These alloys contained respectively 35.7 oz. (alloy No. 3), and 96.04 oz. of gold per ton (alloy No. 5).

The length of time that the alloys were in the furnace and the resulting assays after they were subjected to heating are given in Table VII and Table VIII for the higher temperatures, and in Table IX and Table X for the lower temperatures. The losses at any period of the runs were plotted and gave the curves shown by Plate A. and Plate B. Plate A. shows the losses in ounces of gold per ton of alloy, and Plate B. the percentage loss of gold in terms of the total gold in the original alloy.

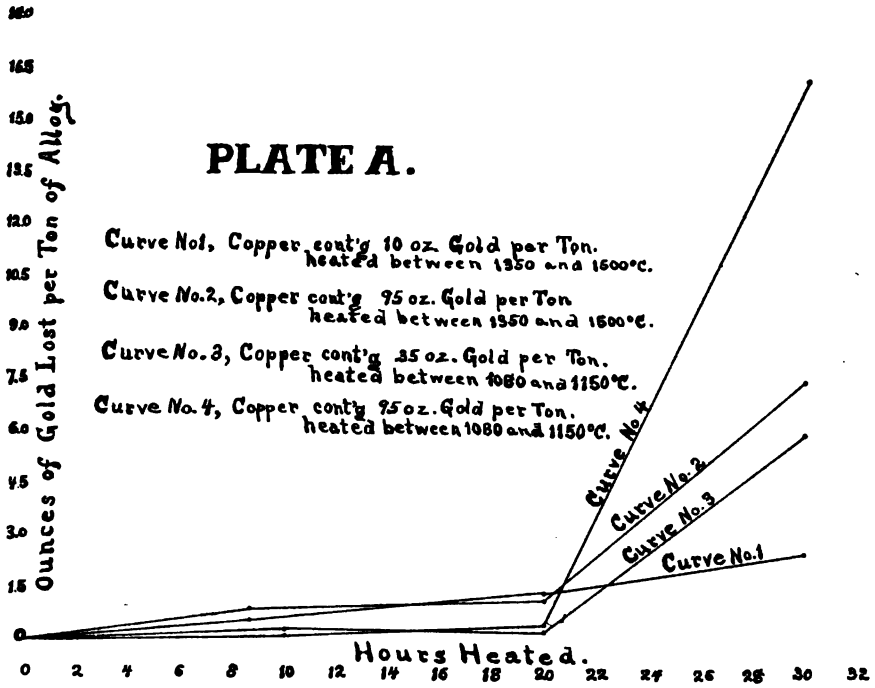


PLATE A.
Curve No1, Copper cont'g 10 oz. Gold per Ton. heated between 1350 and 1500°C.
Curve No.2, Copper cont'g 95 oz. Gold per Ton heated between 1350 and 1500°C.
Curve No.3, Copper cont'g 35 oz. Gold per Ton. heated between 1080 and 1150°C.
Curve No.4, Copper cont'g 95 oz. Gold per Ton. heated between 1080 and 1150°C.
Ounces of Gold Lost per Ton of Alloy.
Hours Heated.
Curve No.4
Curve No.2
Curve No.3
Curve No.1

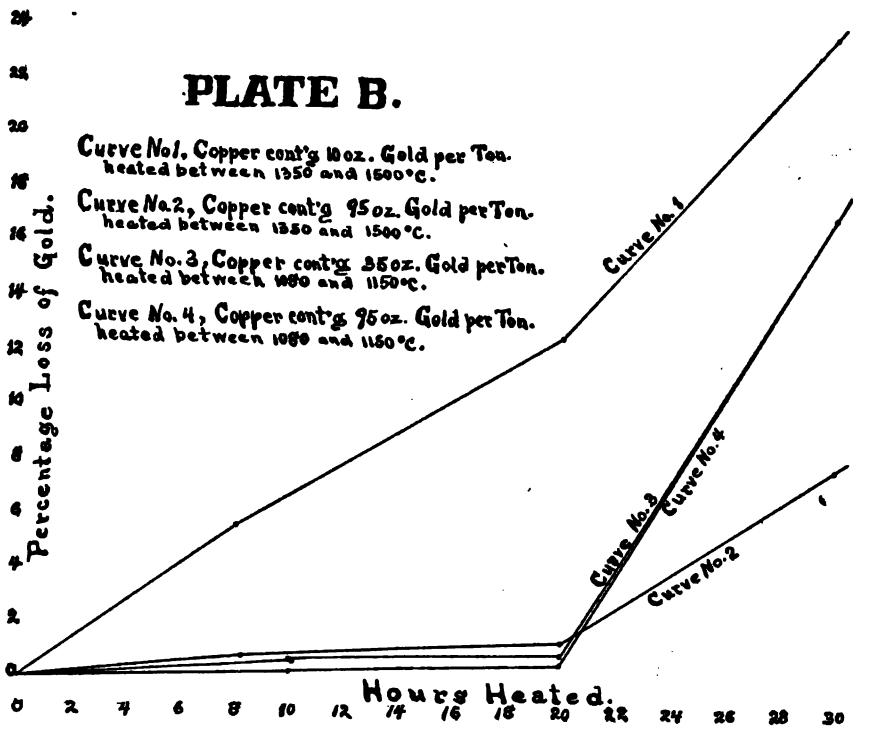


PLATE B.
Curve No1, Copper cont'g 10 oz. Gold per Ton. heated between 1350 and 1500°C.
Curve No.2, Copper cont'g 95 oz. Gold per Ton. heated between 1350 and 1500°C.
Curve No.3, Copper cont'g 35 oz. Gold per Ton. heated between 1080 and 1150°C.
Curve No.4, Copper cont'g 95 oz. Gold per Ton. heated between 1080 and 1150°C.
Percentage Loss of Gold.
Hours Heated.
Curve No.1
Curve No.3
Curve No.4
Curve No.2

TABLE V

RECORD OF HIGHER TEMPERATURE READINGS

1st Day.		2nd Day.		3rd Day.		4th Day.		5th Day.	
Hours Heated.	Temperature °C.	Hours Heated.	Temperature °C.	Hours Heated.	Temperature °C.	Hours Heated.	Temperature °C.	Hours Heated.	Temperature °C.
1	1390	½	1330	½	1330	1	1330	1	1427
2	1440	1½	1440	1½	1380	2½	1427	2	1630
4	1440	3½	1427	3	1340	3½	1440	2½	1330
6	1440	5	1427	5½	1488	5	1440	3	1508
8½	1380			6½	1427	5½	1415	4½	1390
Total time 8½ hours.		Total time 13½ hours.		Total time 20 hours.		Total time 25½ hours.		Total time 30 hours.	

TABLE VI

RECORD OF LOWER TEMPERATURE READINGS

1st Day.		2nd Day.		3rd Day.		4th Day.		5th Day.	
Hours Heated.	Temperature °C.	Hours Heated.	Temperature °C.	Hours Heated.	Temperature °C.	Hours Heated.	Temperature °C.	Hours Heated.	Temperature °C.
1	1103	1	1051	1	1050	1	1110	½	1067
3	1124	2	1145	2	1130	2	1200	1½	1096
5	1105	4	1135	3	1145	2½	1131	3½	1151
7	1110	6	1110	6	1110	3	1151	4½	1124
8	1131	7	1103	7½	1124			5	1110
Total time 8 hours.		Total time 15 hours.		Total time 22½ hours.		Total time 25½ hours.		Total time 30½ hours.	

TABLE VII

ALLOY NO. 2, HEATED BETWEEN 1360 AND 1500°C.

Time of Heating in Hours.	Weight of Sample Taken for Assay.	Gram of Gold in Sample.	Ounces of Gold per Ton of Alloy.	Loss of Gold During Heating Oz. per ton of Alloy.
0	29.1652	0.01044	10.44	
0	29.1594	0.01040	10.40	
0	29.1676	0.01042	10.42	
0	29.1630	0.01040	10.40	0.0
8¼	29.1660	0.00990	9.90	
8¼	29.1500	0.00975	9.76	0.58
20	29.1623	0.00904	9.04	
20	29.1598	0.00892	8.92	1.43
30	29.1544	0.00795	7.95	
30	29.1563	0.00794	7.94	2.47

TABLE VIII

ALLOY NO. 5, HEATED BETWEEN 1360 AND 1500°C.

Time of Heating in Hours.	Weight of Sample Taken for Assay.	Gram of Gold in Sample.	Ounces of Gold per Ton of Alloy.	Loss of Gold During Heating. Oz. per Ton of Alloy
0	29.1672	0.09608	96.07	
0	29.1699	0.09602	96.01	0.0
8¼	29.1605	0.09515	95.16	
8¼	29.1671	0.09522	95.22	0.85
20	29.1598	0.09475	94.77	
20	29.1620	0.09481	94.81	1.28
30	29.1560	0.08840	88.43	
30	29.1594	0.08856	88.56	7.49

TABLE IX

ALLOY NO. 3, HEATED BETWEEN 1080 AND 1150°C.

Time of Heating in Hours.	Weight of Sample Taken for Assay.	Gram of Gold in Sample.	Ounces of Gold per Ton of Alloy.	Loss of Gold During Heating. Oz. per Ton of Alloy.
0	29.1534	0.03568	35.69	
0	29.1511	0.03560	35.62	
0	29.1538	0.03566	35.67	0.0
10	29.1637	0.03546	35.46	
10	29.1568	0.03544	35.44	
10	29.1509	0.03542	35.44	0.21
20	29.1561	0.03534	35.34	
20	29.1531	0.03532	35.33	
20	29.1526	0.03530	35.31	0.33
30½	29.1530	0.02958	29.59	
30½	29.1519	0.02958	29.60	
30½	29.1518	0.02957	29.59	6.07

TABLE X

ALLOY NO. 5, HEATED BETWEEN 1080 AND 1150°C.

Time of Heating in Hours.	Weight of Sample Taken for Assay.	Gram of Gold in Sample.	Ounces of Gold per ton of Alloy.	Loss of Gold During Heating. Oz. per Ton of Alloy.
0	29.1672	0.09608	96.07	
0	29.1699	0.09602	96.01	0.0
10	29.1643	0.09592	95.93	
10	29.1637	0.09592	95.93	
10	29.1665	0.09593	95.93	0.11
20	29.1537	0.09556	95.59	
20	29.1505	0.09553	95.56	
20	29.1538	0.09557	95.60	0.46
30½	29.1534	0.07969	79.73	
30½	29.1515	0.07964	79.68	
30½	29.1549	0.07970	79.73	16.33

METHOD FOR DETERMINING SILVER IN COPPER.

Standard copper-silver alloys were prepared in the same manner as the copper-gold alloys.

Samples of the alloy weighing approximately 5, 10, and 20 grams were used for the assays, the amount taken in each case depending upon the quantity of silver which the alloy contained. The samples were dissolved in concentrated nitric acid (3 c.c. per gram of alloy) in a covered beaker, the excess of acid was evaporated off, and the evaporation continued until copper nitrate commenced to crystallize out of the hot solution. At this point all of the oxides of nitrogen, which are harmful to the precipitation of silver by means of alkali sulphocyanate, were removed. The solution and salts were taken up with 400 c.c. of cold distilled water, and 25 c.c. of a saturated solution of ammonium-sulphocyanate (25 grams per liter) was added to the cold solution as quickly as possible and with constant stirring. The silver in the form of silver sulphocyanate coagulated and settled rapidly to the bottom of the beaker. The solution was filtered and the precipitate washed free from copper salts by distilled water. The filter and precipitate were placed in the original beaker in which the precipitation was made and digested with 80 c.c. of concentrated nitric acid. The beaker was covered with a watch-glass and placed on a hot plate and allowed to remain until all of the paper was decomposed and the precipitate was entirely dissolved. The solution was evaporated to small volume before all of the oxides of nitrogen were driven off, which is essential for the success of the analysis. Evaporation did not extend to dryness. After cooling, the salts were taken up with 300 to 350 c.c. of distilled water, and 20 c.c. of a saturated solution of barium nitrate was added which precipitated the sulphuric acid present:

$$Ba(NO_3)_2 + H_2SO_4 = BaSO_4 + 2HNO_3.$$

The presence of sulphuric acid was the result of oxidation of silver sulpho-cyanate by nitric acid.

About 3 c.c. of a saturated solution of ferric alum, $(NH_4)_2.Fe_2(SO_4)_4.24H_2O$, was added, as an indicator, and the solution titrated at ordinary temperature with a standard solution of ammonium sulpho-cyanate (3.2 grams NH_4CNS per liter).

The standardization was obtained on samples of pure electrolytic silver, dissolved in nitric acid, the solution being boiled to expel nitrous oxide, diluted to about 100 c.c. volume, then titrated. 1 c.c. NH_4CNS solution was equivalent to 0.004465 gram of silver.

$$AgNO_3 + NH_4CNS = AgCNS + NH_4NO_3.$$

The end point is readily distinguished by the sudden appearance of a faint orange color, which the solution assumes as soon as all the silver is precipitated, due to the formation of ferric sulphocyanate.

The accuracy of this method, as described, was first tested on standard copper-silver solutions as follows, and later on copper-silver alloys: About 60 grams of the standard alloy was dissolved in 180 c.c. of concentrated nitric acid in a covered beaker, and the oxides of nitrogen expelled by evaporation. The solution was diluted to 1000 c.c. with distilled water, and three samples of 300 c.c. each were accurately measured out and assayed as already explained. These solutions were titrated with two standardized ammonium sulphocyanate solutions, thus: A solution, 1 c.c. of which was equivalent to 0.004465 gram of silver was run in to within about 1 c.c. of the end-point, then the titration was finished with a second standard solution 1 c.c. of which was equivalent to 0.0005841 gram of silver.

It was thought that by using the two standard solutions, the determinations could be made more accurately. This procedure was, however, found to be unnecessary, so in the later titrations only the stronger solution (1 c.c. equivalent to 0.004465 gram of silver) was used. The results of this series of determinations are recorded in Table XI.

The results of determinations made on the copper-silver alloys are given in Table XII. The analysis was carired out as previously described.

To obtain accurate results by this method of assaying copper-silver alloys, the following precautions should be observed:

If all the oxides of nitrogen are not expelled, and if the solution is too hot when the silver sulphocyanate is precipitated from the copper solution, then the solution will not retain a green color after the excess of ammonium sulphocyanate has been added. In

the first case, the solution will change to the original color, namely blue; at the same time brown fumes of nitrogen tetroxide (N_2O_4) will be given off, causing some of the silver to go back into solu-

TABLE XI

ASSAY OF STANDARD COPPER-SILVER SOLUTIONS

Assay No.	C. C. of Solution Titrated.	Standard NH_4CNS Solution.		Gram of Silver in the Solution.
		1.0 c.c. equivalent to 0.004465 gram silver.	1.0 c.c equivalent to 0.000584 gram silver.	
1	300	18.9	12.0	0.09139
2	300	18.9	12.3	0.09157
3	300	18.9	15.0	0.09145
4	300	19.0	10 8	0.09120
5	300	19.0	10.7	0.09105
6	300	19.0	10.8	0.09120

TABLE XII

ASSAY OF COPPER-SILVER ALLOYS

Assay No.	Alloy No.	Grams of Alloy for Sample.	Standard NH_4CNS Solution. 1 c.c. Equivalent to 0.004465 gram Silver.	Gram of Silver in Sample.	Ounces of Silver per Ton of Alloy.
7	6	20.0279	17.5	0.07814	113.76
8	6	20.0187	17.45	0.077914	113 51
9	7	10.0023	40.4	0.180386	525.99
10	7	10.0036	40.45	0.180609	526 57
11	8	5.0306	50.6	0.22593	1309.90
12	8	5.0213	50.6	0.22593	1311.50

tion. In the second case, on adding the ammonium sulphocyanate to the hot solution, copper is reduced and a precipitate of cuprous sulphocyanate forms which interferes with the accurate titration of the final silver solution.

If the final solution, obtained by dissolving the silver sulphocyanate and the filter paper in concentrated nitric acid is evaporated too far (during the operation of expelling the nitrogen oxide fumes) a brown coating of burnt organic matter from the

filter paper forms on the bottom of the beaker, and when the solution is diluted it assumes a brownish color which interferes with detecting the end point of the titration.

The dissolving of the alloy, the evaporation of the resulting solution, the dissolving of the precipitate, and the final evaporation of this solution should be carefully carried out in a covered beaker, in order to prevent spattering.

VOLATILIZATION OF SILVER.

The heating of the alloys was conducted in a manner similar to the heating of the copper-gold alloys, previously described. Three sets of three alloys were used, alloy No. 6 containing 113.5

TABLE XIII
HEATING OF COPPER-SILVER ALLOYS

1st Day.		2nd Day.		3rd Day.		4th Day.	
Hours Heated.	Temperature °C.	Hours Heated.	Temperature °C.	Hours Heated.	Temperature °C.	Hours Heated.	Temperature °C.
1	1157	1	1175	1	1470	1	1300
3	1224	2½	1340	3	1250	1½	1508
5	1360	4	1415	4½	1470	3	1415
7½	1360	6	1330	5½	1238	4	1405
9	1320	7½	1285	6	1360	5	1415
Total time 9 hours.		Total time 16½ hours.		Total time 22½ hours.		Total time 27½ hours.	

oz. of silver, alloy No. 7 containing 526 oz., and alloy No. 8 containing 1310 oz. per ton. Three 200-gram samples of each were placed in small Dixon graphite crucibles, 3 in. high and 1¾ in. diameter inside, and covered with crushed charcoal. Each crucible was covered with a graphite cover then placed in a graphite box containing charcoal, which was then heated in a gas furnace. As the graphite box held only six crucibles, the heating was so arranged that the alloys which were to be heated for an intermediate period were removd at the end of 16½ hours and replaced by the alloys which were to be heated for the shortest time, 11 hours.

The alloys which were to be heated for the shortest and the longest periods were finished together. The heating was not continuous, but was conducted four successive days, as shown in Table XIII, which contains the tabulated data of the experiments.

Temperature readings were taken at regular intervals with a standard Wanner optical pyrometer. At the end of the heating, each set of alloys was granulated by pouring the molten metal into a tub of cold water, as previously described.

TABLE XIV

HEATING OF COPPER-SILVER ALLOYS

Alloy No.	Time of Heating. Hours.	Average Temperature °C.	Grams of Alloy Taken for Sample.	C.c. of Standard NH_4CNS Solution. 1 c.c. Equivalent to 0.004465 Gram Silver.	Gram of Silver in Sample.	Ounces of Silver per Ton of Alloy.
6	0		20.0279	17.50	0.078140	113.76
6	0		20.0187	17.45	0.077914	113.51
6	11	1400	20 0257	17.45	0.077914	113.48
6	11	1400	20.0374	17.50	0.078137	113.79
6	16½	1350	20.0040	17.40	0.077691	113.27
6	16½	1350	20.0044	17.45	0.077914	113.60
6	27½	1380	20.0610	17.55	0.078360	113.92
6	27½	1380	20.0123	17.50	0.078137	113.88
7	0		10.0023	40.40	0.180386	525.99
7	0		10.0036	40.45	0.180609	526.57
7	11	1400	10 0188	40.50	0.180830	526.16
7	11	1400	10.0188	40.40	0.180380	525.12
7	16½	1350	10.0537	40.60	0.181279	525.93
7	16½	1350	10.0003	40.30	0.179940	524.80
7	27½	1380	10.0492	40.60	0.181279	526.14
7	27½	1380	10.0201	40.40	0.180380	525.06
8	0		5.0306	50.60	0.225930	1309 90
8	0		5.0213	50.60	0.225930	1311.50
8	11	1400	5.0001	50.55	0.225705	1316.58
8	11	1400	5.0158	50.60	0.225930	1313 66
8	16½	1350	5 0420	50.75	0.226598	1310.81
8	16½	1350	5.0675	50.85	0.227045	1307.89
8	27½	1380	5.0102	50.60	0.225930	1315.18
8	27½	1380	5.0187	50.65	0.226150	1314.80

The results of assays of the alloys, before and after heating, are given in Table XIV. In no case was it found that silver was lost by volatilization when copper-silver alloys were subjected to heat above their melting point, and when in contact with charcoal, as was the case with gold when copper-gold alloys were so heated.

Formation of Gold Carbide.

To determine whether gold and copper form carbides, when these metals are heated in contact with carbon, the following experiments were conducted. It was thought that loss of gold might be due to the formation of a carbide which is volatile at a lower temperature than the pure metal.

A sample of 200 grams of copper-gold alloy, containing 95 oz. of gold per ton, was placed in a Dixon graphite crucible, 2½ in. high and 1¾ in. inside diameter. The crucible was completely filled with crushed charcoal, covered with a graphite cover, and placed in another graphite crucible which was also covered.

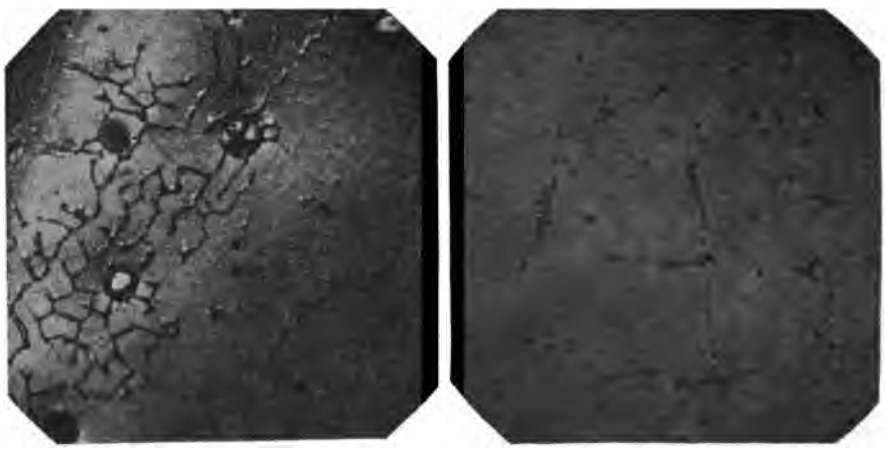
Two samples of pure gold, No. 1 weighing 4.0025 grams, and No. 2 weighing 3.8760 grams, were prepared for heating in the following manner: Sample No. 1 was placed in a small Dixon graphite crucible 1¼ in. high and ¾ in. inside diameter, and covered with crushed charcoal. This crucible was placed in a larger graphite crucible, and this, in turn, inside of a still larger one. All three crucibles were covered with close fitting covers in order to prevent gases and fumes from coming in contact with the metal. Sample No. 2 was placed in a small porcelain crucible (oo), and this put into another large porcelain crucible (o), and both closed with porcelain covers. These were then placed in a covered graphite crucible. No charcoal was in contact with this sample.

The three samples were heated for 28 hours in a gas furnace at a temperature varying between 1340 and 1400° C. Pyrometer readings were taken at regular intervals with a standard Wanner optical pyrometer.

When the porcelain crucibles which contained the gold (sample No. 2) were removed from the furnace, it was found that their covers had firmly fused to them. The crucibles were broken open, and it was seen that fine particles of gold had deposited on the under side of the cover and on the inside walls of the smaller crucible. The nugget (sample No. 2) was weighed and found to have lost 0.0009 gram, or 0.023 percent., showing that gold is volatile at temperatures above 1340° C.

The gold that was heated in contact with charcoal (sample No. 1) was found to have increased 0.0281 gram or 0.7 percent. in weight:

Microphotographs, enlarged 55 diameters, were taken of the unpolished upper surface of the gold (sample No. 2) that had been heated in the porcelain crucible, shown by Plate 1, and also of the condensed particles of gold on the under side of the porcelain cover, shown by Plate 2. The specimens were illuminated by oblique light and exposed for 40 seconds. Plate 1 shows the



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crystallization of gold; the dark network surrounding the individual crystals are cavities, which are the result of the volatilization of gold from the edges of the crystals. Plate 2 shows the condensed gold which collected in the form of skeleton crystals on the under surface of the porcelain cover.

The nugget of gold that had been heated in contact with charcoal (sample No. 1) was cut in two, and one section polished and examined, unetched, under the microscope. It was found that carbon had been absorbed, as is clearly shown by Plate 3, which shows the characteristic gold-carbon eutectic structure. The specimen was illuminated by oblique light, and the magnification was 55 diameters.

In order to substantiate the fact that gold absorbs carbon when it is heated in contact with charcoal above the melting point of the metal, the gold specimen (sample No. 1) was rolled to about 1-16 in. thickness, and its carbon content determined. A sample of 1.2662 gram was digested with 400 c.c. of a solution containing 30 grams of cupric potassium chloride, and 50 c.c. of concentrated hydrochloric acid; a few cubic centimeters of bromine water were added from time to time for the purpose of keeping the copper chloride in the cupric condition, which is necessary for the dissolving of the gold. A 32-c.p. incandescent electric lamp placed under the beaker kept the solution at 60° to 65° C. The gold was dissolved in about ten days, leaving a residue in which the carbon was determined.

The cupric-gold solution was filtered through a weighed Gooch crucible, the residue washed, the crucible, with contents, dried for 16 hours at 80° to 90° C. and weighed. It was then heated in the flame of a blast-lamp for 10 minutes with air access, then for six minutes in an atmosphere of oxygen. The loss of weight by combustion was then ascertained. The carbon content of the gold was 0.7 percent. as the result of heating the metal in contact with charcoal for 28 hours at a temperature of 1340° to 1400° C. and allowing to cool.

The copper-gold alloy, containing 95 oz. of gold per ton, which was heated in contact with crushed charcoal for 28 hours, was cut into three sections. The section cut from the center, weighing 29 grams, was analyzed for its carbon content as follows: The sample was digested with a solution of 200 c.c. of water, 150 c.c. of concentrated hydrochloric acid, and 90 grams of cupric potassium chloride. A few cubic centimeters of bromine water were added from time to time, in order to reoxidize cuprous chloride formed by the reduction of cupric chloride by metallic copper. When all of the metal had dissolved, the solution was filtered through a weighed Gooch crucible. The residue was dried and the crucible with contents was weighed. It was then ignited in the flame of a Bunsen burner and a stream of oxygen was passed over the red hot mass for 10 minutes. As soon as the crucible was sufficiently cooled, it was weighed and found to have lost

0.0101 gram. This loss is the amount of carbon which was present in the copper in the combined and uncombined form. Therefore the sample of copper-gold alloy had dissolved 0.035 percent. of carbon by being heated in contact with charcoal for 28 hours at a temperature varying between 1340° and 1400° C.

A second section of the same copper-gold alloy was polished and etched with dilute (1:1) nitric acid; microphotographs enlarged 55 diameters were taken, as shown by Plate 4. The individual crystals of the copper-gold alloy are clearly shown surrounded by a ground-mass of metallic carbides.

Formation of Copper Carbide.

Although it was evident that gold forms a carbide, and that carbon is dissolved by copper-gold alloy when it is heated above its melting point in contact with charcoal, it was questioned whether a carbide of copper was formed. In order to answer this query, the following experiments were conducted:

A copper-gold alloy (sample No. 3), containing 96 oz. of gold per ton, was placed in a clay annealing cup, and completely covered with sugar carbon, which was prepared by charring granulated sugar. The annealing cup was covered with a porcelain cover and placed in a No. 0000 Dixon graphite crucible. The remaining space in the graphite crucible was filled with sugar carbon, and covered with a graphite top. A sample of pure shot copper (sample No. 4), and a sample of copper-gold alloy containing 70.9 oz. of gold per ton (sample No. 5), were prepared for heat treatment in the same manner. A piece of pure gold (sample No. 6) was placed in a No. 00 porcelain crucible, covered with sugar carbon and a porcelain cover. This was then placed in a No. 0000 Dixon graphite crucible, covered with sugar carbon and a graphite cover.

The four samples, prepared as just described, were heated for 29 hours in a gas furnace at temperatures varying between 1150° and 1450° C. Pyrometer readings were taken every two hours with a standard Wanner optical pyrometer. Records are given in Table XV.

TABLE XV

HEATING OF COPPER-GOLD ALLOYS IN CONTACT WITH CARBON

1st Day.		2nd Day.		3rd Day.		4th Day.	
Hours Heated.	Temperature °C.	Hours Heated.	Temperature °C.	Hours Heated.	Temperature °C.	Hours Heated.	Temperature °C.
1½	1302	1	1360	1	1340	1	1350
3	1380	2	1455	2½	1455	2	1440
6	1415	4	1455	4½	1189	3½	1245
9	1415	6	1455	8	1163	6	1253
Total time 9 hours.		Total time 15 hours.		Total time 23 hours.		Total time 29 hours.	

At the end of 9, 15, 23, and 29 hours the copper and the copper-gold samples (Nos. 3, 4, and 5) were removed from the furnace, cooled, and weighed, to ascertain the loss or gain in weight at these periods. The results of these weighings are given in Table XVI.

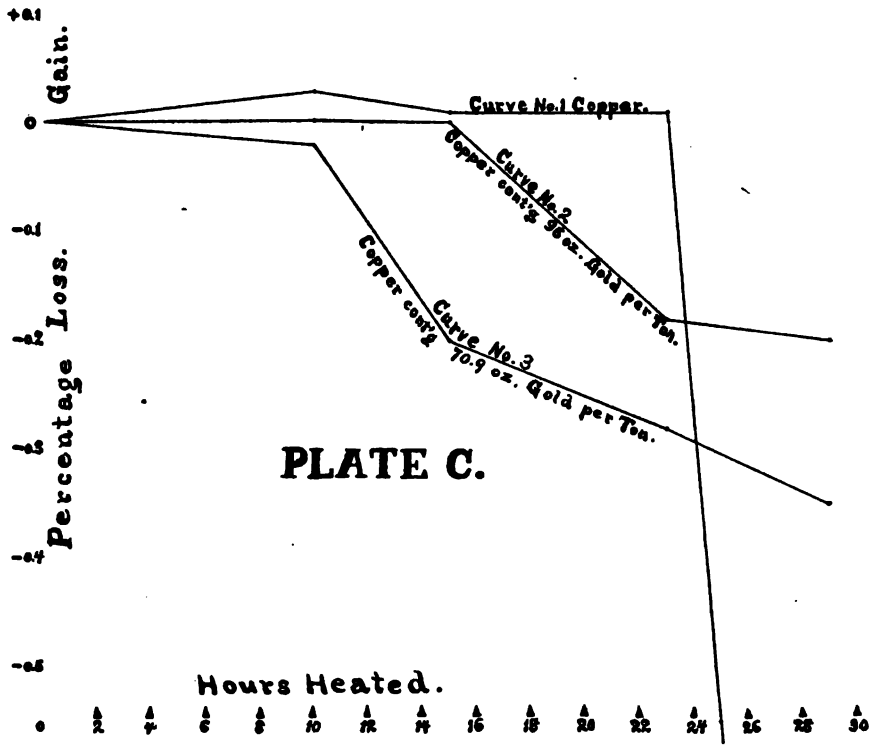
TABLE XVI

RESULT OF HEATING COPPER ALLOYS IN CONTACT WITH CARBON

Sample.	Wt. before Heating.	Wt. after 9 Hrs. Heating.	Wt. after 15 Hrs. Heating.	Wt. after 23 Hrs. Heating.	Wt. after 29 Hrs. Heating.
No. 3; 96 oz. of gold to the ton.....	76.6626	76.6644	76.5532	76.5228	76.5094
No. 4; pure shot copper	75.4360	75.4600	75.4460	75.4500	73.8857
No. 5; 70.9 oz. of gold to the ton.....	71.5522	71.5335	71.4075	71.3480	71.3017

The percentage gain and loss in weight of the pure copper (No. 4) was plotted, giving curve 1 of Plate C. Curves 2 and 3 show the percentage loss in weight of the copper-gold alloys (Samples No. 3 and No. 5) after 15, 23, and 29 hours heating.

Sample No. 6 was weighed only before and after heating for 29 hours. . As some of the gold had fused to the bottom of the crucible, this part was carefully crushed, scorified with 50 grams of test lead, cupelled, and the bead weighed. This was added to the weight of the nugget after heating, which gave the corrected weight and showed a gain in weight of 0.01091 gram, or over 0.7 percent. Fine particles of gold had collected on the



under side of the porcelain cover, showing, as did an earlier experiment, that gold, in the presence of carbon, is volatile at temperatures above its melting point.

The clay annealing cups which held the copper and the copper-gold samples (Nos. 3, 4, and 5) were broken into pieces, when it was seen that they had been impervious to copper during the heat treatment. The porcelain covers were of a brownish-red color, due to the vapors of copper which collected on them and fused

with the glaze. The three samples were cut in two, one of each of the pieces being polished preparatory to taking microphotographs (Plates 5, 6, and 7) and the other pieces being granulated for assay. Table XVII gives the assays of the copper-gold alloys before and after they were heated for 29 hours, in contact with sugar carbon, above their melting point.

TABLE XVII

Sample No.	Time Heated. Hours.	Average Temperature °C.	Weight of Sample Assayed.	Gram of Gold in Sample.	Ounces of Gold per Ton of Alloy.	Loss of Gold per Ton of Alloy.	Percentage Loss.
3	0		29.1672	0.09608	96.07		
3	0		29.1699	0.09602	96 01	0.0	
3	29	1250	10.1759	0.03326	95.08		
3	29	1250	10.2097	0.03328	95.08	0.98	0.98
3	29	1250	10 1516	0.03325	95.08		
5	0		10.0057	0.02438	70.87		
5	0		10.0307	0.02440	70.87	0.0	
5	29	1250	10.0081	0.02408	70.17		
5	29	1250	10.0398	0.02410	70.00	0.77	1.08
5	29	1250	10.0790	0.02402	69.82		

Plate 5 is a microphotograph of the copper (sample No. 4) which had been heated in contact with sugar carbon at temperatures between 1150° and 1450° C. for 29 hours. The specimen was polished, etched with dilute (1:1) nitric acid, and exposed for 75 seconds, illuminated with oblique light, and magnified 45 diameters. It shows crystals of copper surrounded by a ground-mass of the copper-carbon eutectic.

Plate 6 is a microphotograph of copper containing 96 oz. of gold per ton (sample No. 3) after being heated in contact with sugar carbon for 29 hours at temperatures between 1150° and 1450° C. The specimen was polished, etched, and photographed with an exposure of 35 seconds and a magnification of 35 diameters. The individual crystals of the copper-gold alloy are shown surrounded by a heavy ground-mass of the carbides of the metals.

Plate 7 is a microphotograph, enlarged 35 diameters, of copper containing 70.9 oz. of gold per ton (sample No. 5) heated in the

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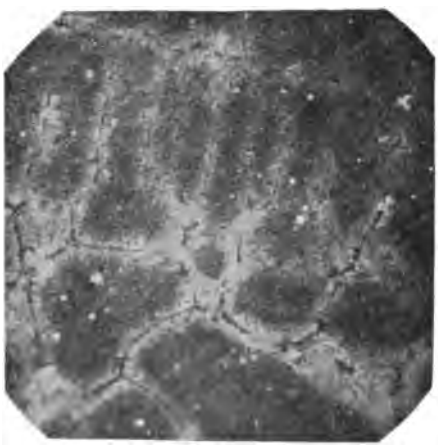

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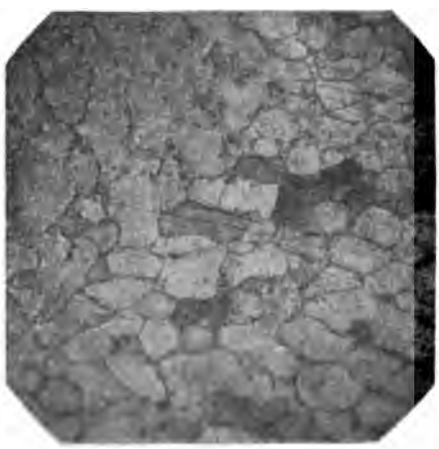

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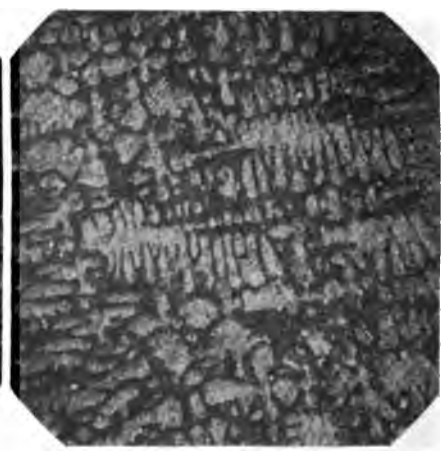

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same manner as Samples No. 3 and No. 4. The specimens were photographed with an exposure of 45 seconds and a magnification of 35 diameters. The crystals of the copper-gold alloy are shown with a thin ground-mass of the carbides of the metals.

Plate 8 is a microphotograph of the gold (sample No. 6) which had been heated in contact with sugar carbon for 29 hours at temperatures between 1150° and 1450° C. The specimen was polished and etched electrolytically in a 3-percent. potassium cyanide solution. It has a magnification of 45 diameters, being exposed for 45 seconds with oblique light illumination. The gold-carbon eutectic is clearly shown by the characteristic dentrites, the same structure as is shown by Plate 3, but was brought out more clearly by the electrolytic etching. These two plates, together with the analysis, prove the existence of an alloy of gold and carbon, the eutectic being formed by prolonged heating of gold in contact with carbon above the melting point of the metal.

Comparing Plates 6 and 7, which are of the same magnification (35 diameters), it is seen that in the alloy containing 96.08 oz. of gold per ton there is a larger amount of that constituent which surrounds the individual crystals than in the alloy which contains 70.9 oz. of gold per ton. This shows that the ground-mass, which is principally gold carbide, varies as the gold content of the alloy. The same structure is shown by Plates 4 and 5, the magnification of these latter plates being 55 and 45 respectively.

CONCLUSIONS.

Segregation of copper-gold and of copper-silver alloys can be prevented by granulating them, as was proved by obtaining checked results on duplicate and triplicate assays.

Silver was not lost by volatilization when samples of copper, containing respectively 113 oz., 526 oz., and 1311 oz. of silver per ton, were heated in contact with carbon, above the melting point of the metal, for 27½ hours.

Pure gold was found to volatilize at 1350° C., when heated for 28 hours in covered porcelain crucibles.

Gold was found to volatilize from copper, containing 10 to 100 oz. per ton, when heated in contact with carbon at temperatures

between 1130° and 1450° C. The losses which occurred when heated less than 10 hours were small, but after 14 hours heating the losses were high and increased rapidly with further lapses of time. The reason for this is porbably the formation of carbides of gold which are volatile at temperatures above 1100° C.

Copper slowly absorbs carbon when heated in contact with it at temperatures above the melting point of the metal. After the copper is saturated with carbon, a rapid loss of metal occurs, as the result of the volatilization of a copper carbide; the loss is insignificant under 10 hours heating, is moderate between 10 and 23 hours, but is rapid after 23 hours of heating.

Gold forms an eutectic when heated, in contact with carbon, above its melting point for a long period of time. Samples of gold heated in contact with carbon for 28 hours above 1100° C. showed the characteristic dentrites of the eutectic structure. The analysis of these samples, which were heated and cooled in contact with carbon, showed them to contain over 0.7 percent. of carbon. The carbide of gold was found to be volatile above the melting point of the metal.

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THE CAUSES AND THE PREVENTION OF COAL MINE EXPLOSIONS.

BY EDWARD K. JUDD.[1]

The causes of coal mine explosions are now fairly well understood, and most of the larger operators have instituted precautions to avoid them, in addition to the provisions required by the mining laws of the several states. Public attention, however, is still forcibly arrested at too frequent intervals by some wholesale disaster, involving the loss of scores of lives, and yet it is not these that contribute most to the total misery; they are, so to speak, only the punctuation points in a continuous narrative of death and injury, of little mishaps occurring every other day, taking toll of only one or two lives each time. In this respect, the United States, allowing every year four men to be killed out of every 1000 employed in coal mining, falls behind all other important countries. In Great Britain the average is 1.3, while in Belgium, noted for the stringency of its mining regulations, only one man is killed anually out of 1000 employed. Death in coal mining results from many and varied causes, but fires and explosions do not head the list; of all the coal miners killed in the United States in 1909, about 45 percent. had coal or roof rock fall on them, while only 14 percent. were the victims of mine explosions. The statistics indicate, however, that an explosion is much more deadly than any other form of accident, in that it is more likely to kill its victims outright than merely to maim or injure them.

The following discussion will be confined to coal mine explosions proper, essentially gas explosions, as distinguished, on the one hand, from mine fires, and, on the other, from explosions of dynamite, either of which catastrophies might happen at any time in any kind of mine. It may be stated in passing that the smoke of burning wood or coal and the fumes of slowly burning dyna-

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mite are nearly as much to be dreaded underground as the more instantaneous gas explosion.

The atmosphere is the medium in which explosions are generated and transmitted in a coal mine. The air circulating in a coal mine differs from that above ground in having less than the normal percentage of oxygen but relatively high percentages of certain explosive and noxious gases. Of these, the most prevalent and also the most dangerous is methane, or "firedamp." This is a light, inflammable gas resembling ordinary illuminating gas, except for its lack of odor. It appears to be present more or less in all coal beds, and as the mine workings penetrate and radiate out into a coal seam, the firedamp exudes into the mine atmosphere. Even after all the marketable coal has been taken out of a mine, vast quantities of slaty coal are generally left behind, and this continues to exhale the gas. If other seams exist above or below the one under operation, the gas in them is often free to migrate through the innumerable small fissures inevitably produced by the process of mining. Like other inflammable gases, if mingled with a proper amount of air, say 10 percent. of gas to 90 percent. of air, firedamp forms an explosive mixture which may then be ignited by a variety of means, with disastrous results.

Another ingredient found in unnatural proportions in coal mine air is carbon dioxide, or "blackdamp." This is a heavy, non-poisonous and odorless gas, the important characteristic of which is that it will extinguish flame and smother life with equal promptness and efficacy. It is formed in large quantities whenever any carbonaceous material, wood, coal, oil or gas, is burned in an ample supply of air, as in a stove or a lamp. It is also given off by the respiration of men and animals, and by the explosion of most blasting powders. Under ordinary working conditions, the amount of blackdamp in mine atmosphere is not dangerous, though sometimes oppressive, but if a vigorous fire is raging underground, or if an explosion of firedamp has just occurred, blackdamp will flood the lower levels of the mine, suffocating every form of animal life.

Another noxious ingredient of mine air is carbon monoxide, familiar to everybody as the pungent, blue-burning, "coal gas." If inhaled repeatedly, even though greatly diluted, its effect is

fatal. It is generated mainly by the imperfect combustion of carbon resulting from an insufficient supply of air. Fortunately it does not occur in dangerous amounts under ordinary working conditions, but, like blackdamp, is formed in great quantities whenever a fire or explosion happens underground.

The importance of fine coal dust, which is inevitably present in mine air, as a contributing factor to colliery explosions has only lately been fully realized. In a state of fine division, and intimately mixed with air, coal dust forms a most violent explosive compound; the dust of soft, bituminous coal is more dangerous in this respect than that of anthracite. This phenomenon has been the subject of careful scientific investigation by the governments of European countries, and by the United States Bureau of Mines, which maintains a laboratory for the purpose at Pittsburg. If firedamp is also present in a cloud of coal dust, the violence of an explosion is still further enhanced.

The most satisfactory method of robbing coal dust of its dangers is to lay it, and keep it laid, by the judicious and frequent sprinkling of an ample quantity of water. It is recommended that the face, walls and roof of a mine working place should be thoroughly drenched by a hose before setting off a blast to break down the coal, thus counteracting the evil at its principal source. In Germany they automatically sprinkle the loaded coal cars to prevent dust from blowing off from them as they travel through the mine. Some mines seek to maintain a certain degree of universal dampness by injecting into the ventilating current all the available exhaust steam from the mining plant. Others maintain the roadways in a moist condition by scattering quantities of some salt, like calcium or magnesium chloride, which has the property of absorbing water out of the air and retaining it. The use of water underground for laying coal dust has the disadvantage that, when sprinkled in the profusion demanded for perfect safety, it frequently causes the floor or the roof of the coal seam to slack and swell and the timbers to rot, while also producing a condition of the atmosphere very disagreeable and unsanitary for those who have to work in it. Instead of water, fine clay dust, or sand, is sometimes spread thickly along the passageways, diluting the coal dust and thereby depriving it of com-

bustibility. Coal dust is almost certainly the cause of those explosions that occur now and then in mines reported and commonly supposed to be free from gas, of which the Ott No. 20 mine disaster is a recent example. It is doubtful whether any of the precautions above referred to would suffice to extinguish a dust explosion once well under way, but they would be efficient if applied in those parts of a mine in which an incipient explosion is to be feared.

The dangers peculiar to coal mining are thus seen to depend mainly upon the condition of the underground atmosphere; whence it follows that the most universal precautionary measure is to ventilate the mine workings with a copious circulation of fresh air from outside. To maintain such a current of air through an intricate network of radiating, intersecting and steadily lengthening passages, enclosing often an area of many square miles, is the most difficult problem encountered in coal mining. In a general way, the main passages of a coal mine are driven in pairs, separated by "pillars" of solid coal not less than 25 feet and often 100 feet thick. At least two separate openings are made to the surface, down one of which fresh air enters, travels along one side of the pillar, circulates through the workings and finally emérges from the other opening. A huge centrifugal fan, driven by steam engine or electric motor, is placed at one of the openings to create the circulation. Fans most commonly exhaust the bad air out of a mine, but sometimes the direction is reversed, the fan acting then as a blower; the change can usually be made in a few minutes by a simple shifting of swinging doors, without stopping or reversing the rotation of the fan. The amount of fresh air required depends on the gas-making proclivities of the particular mine, and is also proportioned to the number of men and horses employed underground; the laws of Pennsylvania advise 100 cubic feet of fresh air per minute for each man, and 500 cubic feet for each horse or mule in mines not thought to be excessively gassy. It is one of the duties of mine inspectors to regulate this matter.

It has long been suspected that sudden fluctuations of barometric pressure exert a powerful influence on the rate at which a given mine will yield its flow of firedamp, and a number of

German engineers have recently demonstrated the truth of this suspicion to a practical certainty. It seems only reasonable that while the barometer stands high, indicating abnormally great atmospheric pressure, the gas issuing from a coal seam, which it does under relatively enormous pressure, will be constrained to retire into the cavities that exist in great volume among the old workings of a mine; when the barometer falls, indicating that the atmospheric pressure has been relieved, this accumulation of gas is then free to escape more or less suddenly into the mine air, in a volume with which the fan is momentarily unable to cope. As a means of counteracting this danger, it has been suggested that while the barometer stands low the fan, operated as a blower, should be speeded, so as to maintain the air pressure underground more nearly constant.

So much for the conditions that make an explosion possible; what next are the means by which it is actually ignited? A common offender is the naked, unprotected flame of the oil lamp worn on the miner's cap. Such lamps are not supposed to be used inside a mine that is gaseous, but miners sometimes disobey instructions; moreover, a sudden burst of gas has been known to occur in a mine theretofore nearly free from it, in which naked lights have been habitually in use. This phase of danger long ago led to the design of the "safety lamp," the fundamental principle of which is that a flame will not at once penetrate a fine wire gauze. In the modern safety lamp a heavy glass cylinder surrounds the flame, for the sake of better illumination, but the upper part of the flame chamber is surrounded by one or two wire gauze jackets. If then the lighted lamp be carried inadvertently into an explosive atmosphere, this enters and burns harmlessly inside the lamp; after a time the gauze would become red hot and might possibly cause an explosion of the gaseous atmosphere outside, but before this can happen the miner is expected to extinguish the lamp or to retreat quickly out of the dangerous atmosphere. Incidentally, the presence of firedamp and its approximate percentage can be detected by noting the behaviour of the flame inside a safety lamp; the richer the air is in firedamp the higher will be the blue capping on the yellow flame of the lamp. It is the duty of the "fire-boss" to make a tour

of all the working places in a mine, carrying a delicate safety lamp, before the next shift goes to work, and to bar all miners out of places that he considers unsafe. Strange as it may seem, most of the modern improvements in the design of safety lamps have been directed towards making it more difficult for a miner to open his lamp underground, whether to obtain better illumination or to light his pipe. All manner of ingenious locks have been devised to prevent thoughtless miners from thus risking their own and their neighbors' lives.

Another frequent occasion of explosions is the "blown-out" or "windy" shot. After having undercut the block of coal it is desired to break down, the miner bores two or more holes into the face of the coal close to the top of the seam. These are then charged with explosive and fired. For this work a powerful explosive is not so desirable as a slow-burning one like black powder, which will exert a rending and heaving pressure rather than a shattering force. Low-grade dynamites are also used, having the advantage of not yielding a thick smoke. If the miner should load more powder than necessary into the hole, or should not tamp it carefully with a sufficient amount of clay, on exploding the charge burning grains of powder would be ejected from the hole and would ignite any explosive mixture of gas or dust that might be present. Such an accident is particularly likely to occur if the miners are allowed to indulge the lazy habit of "shooting off the solid," that is, firing shots without first properly undercutting the coal; this practice is not only dangerous but it pulverizes and wastes the coal, and is now generally condemned.

To diminish the danger from this source, a great variety of "safety explosives" have been compounded, the uniform characteristic of which is that they produce a minimum amount of flame and even that is quickly extinguished by the gaseous products of the explosion itself. As a further means of limiting the danger from windy shots, "shot firers" are employed in some mines, whose duty it is to direct and inspect the charging of blast holes and then to explode them after the other miners have retired to safety. With the idea of obviating the use of explosives altogether, a very safe precaution in a notably gaseous mine, a device known as the "hydraulic cartridge" has recently been introduced

into this country; this operates on the principle of the hydraulic press and is worked by hand, breaking down the block of coal after it has been undercut.

When electric power began to be largely utilized in coal mining, it was apprehended that the sparks inevitably caused thereby would be a fertile source of mine gas explosions. Opinion differs as to the actual likelihood of such an event, but in order to be on the safe side, extra precautions are always observed in the installing of electric systems underground. Electric locomotives, taking power from a trolley wire, are generally interdicted in notoriously gaseous mines and in their place compressed-air locomotives are commonly used. The invention of the alternating-current induction motor, which, having no commutator can make no sparks, was a great blessing to coal operators, and motors of this type are now largely in use.

The commonly observed precautions against explosions have been referred to above, in describing the sources of the danger. The coal mining regulations of the several states are more or less excellent but great difficulty seems to be encountered in enforcing them rigidly and impartially. If the actual prime causes of a long list of mine explosions were tabulated it would be noticeable that a majority of them were occasioned by some reckless breach of rules by a thoughtless or ignorant miner. Operators, too, have often been negligent in anticipating disasters and in providing means for escape or rescue. It is now generally realized that one of the greatest obstacles to an effective enforcement of the mining laws lies in the fact that the State Mine Inspectors generally are not appointed by the executive or the judicial authorities, but are elected by popular vote. The embarrassments under which an inspector must labor can easily be imagined, who, owing his election to the votes of the operators or of the miners, and hoping for re-election by one or the other of them, is called upon to exercise his soundest judgment in a matter of dispute between them. Then finally, the labor unions would earn well deserved praise and would strengthen their own cause if they would of their own accord discipline their members when found guilty of dangerous practices, instead of upholding the culprits, as is their more habitual procedure.

HEAT TREATMENT OF A NICKEL STEEL.*

BY W. CAMPBELL † and H. B. ALLEN.††

Nickel steel has received a good deal of attention, notably by D. H. Browne[1] and A. L. Colby.[2] Stoughton, in his "Metallurgy of Iron and Steel," states that 3.5 percent. of nickel added to carbon steel will increase the elastic limit nearly 50 percent., while reducing the ductility by only 15 to 20 percent. About 3.5 percent. of nickel will give steel approximately six times the life in resistance to fatigue. Under the head of "Nickel Steel — Forging Quality," the following figures are given:

Composition: C, 0.25 to 0.3; Mn, 0.5; Ni, 3.2 per cent.
Elastic limit: 49,270; 57,300 lb. per sq. in.
Tensile strength: 87,360; 77,800 lb. per sq. in.
Elongation in 2 in.: 34; 31.5 per cent.
Reduction of area: 58; 62 per cent.

Guillet,[3] in a paper on "Steels used for Motor Car Construction in France," gives the following figures:—

Nickel percent.	Carbon percent.	Condition.	Tensile strength Tons.	Elastic limit Tons.	Elongation percent.
1 to 2	0.3 to 0.4	As rolled	31.75 to 38	20 to 24	27 to 20
		Quenched at 800°C, Annealed at 500°C	44.5 to 57	35 to 47	16 to 12
3.25 to 3.5	0.2 to 0.25	As rolled	35 to 41	21.5 to 25.5	25 to 20
		Quenched at 800°C, Annealed at 500°C	54. to 66	42 to 54.5	14 to 11

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†† Met. E., Columbia, 1911.
[1] *Transactions,* American Institute of Mining Engineers, Vol. XXIX, 1899. p. 569.
[2] "A Comparison of Certain Physical Properties of Nickel and Carbon Steel," 1903, Bethlehem Steel Company.
[3] *Journal,* Iron and Steel Institute, 1908, II, p. 177.

Waterhouse[4] studied the heat treatment of a series of steels with 3.8 percent. nickel and 0.41 to 1.52 percent. carbon, and found that they behaved the same as ordinary carbon steels as regards temperature and microstructure. Overheating was quite perceptible between 1100° and 1200° C. As compared with ordinary carbon steels, the critical points, Ar_{1-3}, are 20° lower for each percent. of nickel.

Souther[5] recommends for a steel of 3.3 to 3.6 percent. nickel, 0.2 to 0.3 percent. carbon, and 0.5 to 0.8 percent. manganese, the following heat treatment: "After forging, heat to 844° C. and quench in oil. Reheat to 426° to 537° C. and cool slowly."

In an annealed condition, this steel will have an elastic limit of about 60,000 lb. per sq. in. and a reduction in area of over 50 percent. After heat treatment an elastic limit of 90,000 lb. per sq. in. and a reduction in area of over 50 percent. is obtained. This strength can be increased by quenching in water or in brine and by varying the drawing temperature. It is possible to obtain an elastic limit of 120,000 lb. per sq. in. with at least 40 percent. reduction of area. The endurance of steel so treated, under vibration or alternate stress, is at least ten times that of a steel in an annealed condition.

The material used in the present research was open-hearth steel, containing carbon, 0.27; nickel, 3.15; manganese, 0.65 percent.; rounds, 5/8-in. diameter; cold drawn. The investigation included:

1. Determination of critical points and of refining temperature (by microstructure).
2. Heating to various temperatures from 600° to 1350° C. and cooling (*a*) in air; (*b*) in furnace; (*c*) by quenching in water; (*d*) by quenching in oil.
3. Heating to temperature giving maximum strength when quenched (900° C.) and (*a*) quench in water; (*b*) quench in oil, then draw at 200° to 660° C. and cool in air.

The physical properties, elastic limit, tensile strength, elongation in 2 in., and reduction of area, were determined under (2) and (3), and the microstructure of the series was compared.

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[4] *Proceedings*, American Society for Testing Materials, Vol. VI, 1906, p. 247.

[5] Society of Automobile Engineers, 1910.

Determination of Critical Points.

A piece of steel of 5/8-in. diameter and 1 in. long was used, with a small hole drilled to center. The time was recorded by stop-watch and the temperature by a platinum-platiniridium couple and Siemens-Halske galvanometer. The heating was done in an electric furnace.

Ac_1	begins 670° C.;	maximum 695°C.	
Ac_{2-3}	begins 725 ;	maximum 750.	Ends 780° C. (?)
Ar_3	begins 795 (?) ;	maximum 760	
Ar_2	begins 725 ;	maximum (705) 660	
Ar_1	begins 630 ;	maximum 590.	

To check the point Ac_{2-3}, or the refining temperature, a piece of steel was heated to about 1300° C. and cooled in the furnace. Pieces were reheated to 680°, 710°, 720°, 750°, 790° C. etc., and slowly cooled. Under the microscope the piece heated to 720° still showed traces of overheating, while that heated to 750° was refined. The piece heated to 790° showed a coarser grain.

Hence we may say that Ac_{2-3} is complete at 750° C. which is about 75° lower than for ordinary carbon steel.

Heat Treatment.

The test bars were heated in a gas forge, using a double iron muffle to insure uniform heating. The temperature was controlled by a thermo-couple inclosed in a quartz tube. The time of heating was from 35 to 50 minutes, and the bars were held at the maximum temperature for five minutes to insure uniform conditions. The furnace was then opened and three bars were withdrawn; one was cooled between two bricks in air, another was quenched in about 30 gal. of water at room temperature, and the last was quenched in oil. The furnace was then closed and the fourth bar allowed to cool there.

In Table I are given the physical properties of the bars heated to various temperatures and cooled in air. At the bottom of the list is No. 16, the steel as received. Reheating to 700° C., *i.e.*, above Ac_1, decreases the elastic limit as shown in No. 16_{11}, while the increase in elongation is very marked, as we should expect. The next heat, No. 16_{14}, to 750° C. (Ac_{2-3}), gives the maximum ductility. At 1200° C. (No. 16_{29}) there is a marked increase in

TABLE NO. I
BARS HEATED TO DIFFERENT TEMPERATURES AND COOLED IN AIR

No.	Cooled in air from—°C.	Yield point, lb. per sq. in.	Ultimate strength, lb. per sq. in.	Elongation in 2 in. percent.	Reduction in Area, percent.
16_{11}	700	66,630	90,370	33.3	62.8
16_{14}	750	67,400	89,500	37.5	64.4
16_{17}	800	63,600	89,000	34.5	50.5
16_{20}	900	59,660	86,500	35.0	58.7
16_{23}	1000	58,240	87,840	33.0	59.7
16_{32}	1100	59,000	88,300	32.5	60.5
16_{29}	1200	71,740	93,340	25 0	42.3
$17_{40.2}$	1250	63,360	93,400	22.5	35
$17_{47.2}$	1300	69,700	95,500	...	36.2
$17_{48.2}$	1350	77,200	98,400	10	11.5
16	as received.	83,170	93,175	22	58.7

strength and a corresponding falling off in ductility. This becomes very marked at 1350° C. (No. $17_{48.2}$).

In Table II are given the physical properties of the bars cooled

TABLE NO. II
BARS HEATED TO DIFFERENT TEMPERATURES AND COOLED IN FURNACE

No.	Cooled in furnace from —°C.	Yield point, lb. per sq. in.	Ultimate strength, lb. per sq. in.	Elongation in 2 in. percent	Reduction in area, percent.
16_{36}	610	71,650	91,100	32.0	61.0
16_{13}	700	60,350	86,700	36.0	59.0
16_{15}	750	61,400	86,700	39.5	58.8
16_{18}	800	.....	85,300	37.0	54.8
16_{21}	900	53,240	84,710	34.5	59.5
16_{24}	1000	52,490	85,760	33.0	55.1
16_{27}	1100	51,650	84,400	35.0	59.4
16_{30}	1200	51,390	83,800	35	54.9
16	as rolled.	83,170	93,175	27	58.7

in the furnace. Heating to 610° C. (No. 16_{36}) lowers the elastic limit and increases the ductility, because heat treatment even below the critical points has marked effects on the properties of material that has been worked cold. As before, heating to 750° C.

gives the most ductility, but even the 1200° C. heat shows little difference, showing that slow cooling may mask overheating.

The physical properties of the bars quenched in water are given in Table III. The sharp jump between No. 16_{38} and No. 16_{10}

TABLE NO. III

BARS HEATED TO DIFFERENT TEMPERATURES AND QUENCHED IN WATER

No.	Quenched from—°C. in water.	Yield point lb. per sq. in.	Ultimate strength, lb. per sq. in.	Elongation in 2 in., percent.	Reduction in area, percent.
16_{35}	610	81,300	101,000	27	54.5
16_{37}	665	72,370	94,480	25.5	64.0
16_{38}	680	67,240	103,950	.....	.....
16_{10}	700	136,900	153,200	.....	14 0
16_{13}	750	197,000	211,200	2.5	4.4
16_{16}	800	223,000	223,000	1.0	2.4
16_{39}	820	146,450	230,650	2.0	4.5
16_{28}	850	139,100	162,800	.....	.....
16_{40}	870	168,360	275,800	3.0	7.0
17_{48-1}	895	.....	252,300	2 5	4.1
16_{19}	900	112,200	233,800	.....	.....
17_{22}	900	159,380	268,150	2.0	7.9
16_{24}	900	118,200	253,780	2.5	5.3
17_{44-1}	925	.....	245,200	4	6.8
16_{41}	945	206,400	265,000	2.5	5.5
16_{45-1}	980	.....	228,500	2	3.7
16_{22}	1000	147,500	164,300	0.0	4.4
17_{46-1}	1000	.....	252,000	6.5	.....
16_{25}	1100	.....	230,900	.....	.....
16_{26}	1200	.....	234,000	.....	.....
17_{49-1}	1250	.....	155,000	0	4.9
17_{47-1}	1300	.....	96,400	.....	1.0
16	as rolled.	83,170	93,175	27	58.7

shows that at 700° C. Ac_1 has been passed, thus confirming the other determinations. The maximum tensile strength of 275,800 lb. per sq. in. occurred in the bar heated to 870° C. (No. 16_{40}). The tensile strength remains high to 1200° C. heat. The results on quenched bars are liable to vary and are uncertain, as seen by comparing the three heats to 900° C. This is due in part to the slight set formed in the quenched bars at times. However, there are enough data to show that the strength does not fall off immediately above the maximum, as has been asserted.

Table IV gives the physical properties of the bars quenched in oil. The tensile strength is high, but somewhat below that of the water-quenched bars. The elastic limit shows a falling off, which may be accidental.

TABLE NO. IV

BARS HEATED TO DIFFERENT TEMPERATURES AND QUENCHED IN OIL

No.	Quenched from—°C in oil.	Yield point lb. per sq.in.	Ultimate strength, lb. per sq. in.	Elongation in 2 in., percent.	in area, Reduction percent.
17_{42-2}	895	197,300	236,000	10	29
17_{24}	900	161,400	202,730	12.5	51.2
17_{44-2}	925	162,800	229,300	10	26
17_{45-2}	980	119,700	231,400	8.5	23.6
17_{42-2}	1000	99,200	235,200	10.0	24.2

Bar 17_{24} quenched at 900° shows such an abnormal reduction of area that it cannot be correct. It is probable that it was taken from the oil too soon. Bar $17_{43.2}$ is therefore taken as the standard in Table VI.

Reheating Tests.

The next part of the work was the reheating of quenched bars; and the question arose as to the best temperatures for the initial quenching. Looking over the results shown in Fig. 1, we see that there is marked similarity between the air- and furnace-cooled bars as regards tensile strength and elastic limit, the air-cooled in general being the stronger. This fits in with carbon steels generally. The ductility of the furnace-cooled bars is just a little greater, but the difference in all properties is not so marked as in carbon steels.[6] The falling off at 1200 ° C. shows overheating.

In regard to the quenched bars, Ac_1 is well marked on the tensile-strength curve and by the drop in ductility at 700° C.

[6] W. Campbell: *Proceedings*, American Society for Testing Materials, Vol. VII, 1907, p. 240.

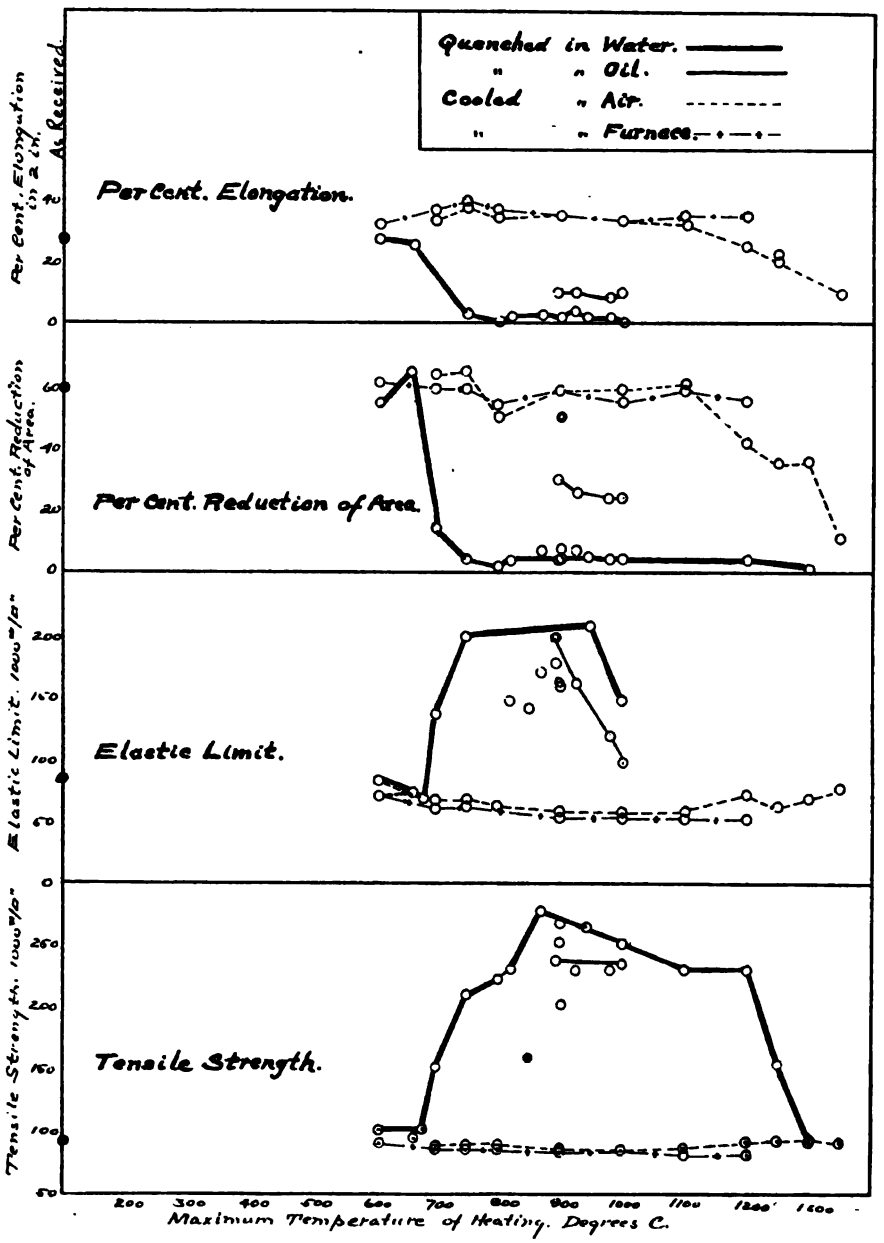
Quenched in Water.
" " Oil.
Cooled " Air.
" " Furnace.
Per Cent. Elongation in 2 in. As Received.
Per Cent. Elongation.
Per Cent. Reduction of Area.
Per Cent. Reduction of Area.
Elastic Limit. 1000 #/in²
Elastic Limit.
Tensile Strength. 1000 #/in²
Tensile Strength.
Maximum Temperature of Heating. Degrees C.

FIG. I.

The upper point, Ac_{2-3} at 750° C., would be expected to show nearly the maximum strength and minimum ductility. The elastic limit of this heat is high, but the maximum for tensile strength is at 870° C. This is most probably due to the rate of cooling in the quenching process. It may be that we have to heat well above the critical point to get the maximum effect.

In regard to the best temperature for initial quenching of the material to be tempered, the choice lay between just above Ac_{2-3}, say 760° C., and 870 to 900° C., where the maximum strength was obtained. The latter temperature was chosen because, among other reasons, the grain would be much coarser than at 760°, whence the process of tempering could be followed more easily in studying the miscrostructure.

Six bars were heated to 900° C., quenched in water, then reheated to 200°, 300°, 400°, 500°, 600° and 650° C. in an electric furnace, and finally cooled in air. The time of tempering was about 45 minutes at maximum temperature. Six bars which had been quenched in oil from 900° C. were similarly treated. The figures for the two sets are given in Tables V and VI, and plotted in Fig. 2.

TABLE NO. V

BARS HEATED TO 900° C., QUENCHED IN WATER AND REHEATED TO VARIOUS TEMPERATURES

No.	Reheated to —°C.	Yield point, lb. per sq. in.	Ultimate strength, lb. per sq.in.	Elongation in 2 in., per-cent.	Reduction in area, percent.
17₂	0	159,380	268,350	2	7.9
17₂₋₆	150	206,000	230,000	15	45
17₂₋₇	175	183,500	219,600	15	47
17₂₋₆	200	196,000	233,500	12	54.0
17₂₋₁	300	166,140	189,070	14	58.8
17₂₋₂	400	139,700	153,730	15.5	59.9
17₂₋₃	500	112,580	126,690	19	62 7
17₂₋₄	600	91,815	105,160	26	68.5
17₂₋₅	650	70,300	93,760	30	69.0
16	as received.	83,170	93,175	27	58.7

Water-quenched and tempered.— The original quenching gives a low ductility, elongation 2 percent., reduction in area about 8

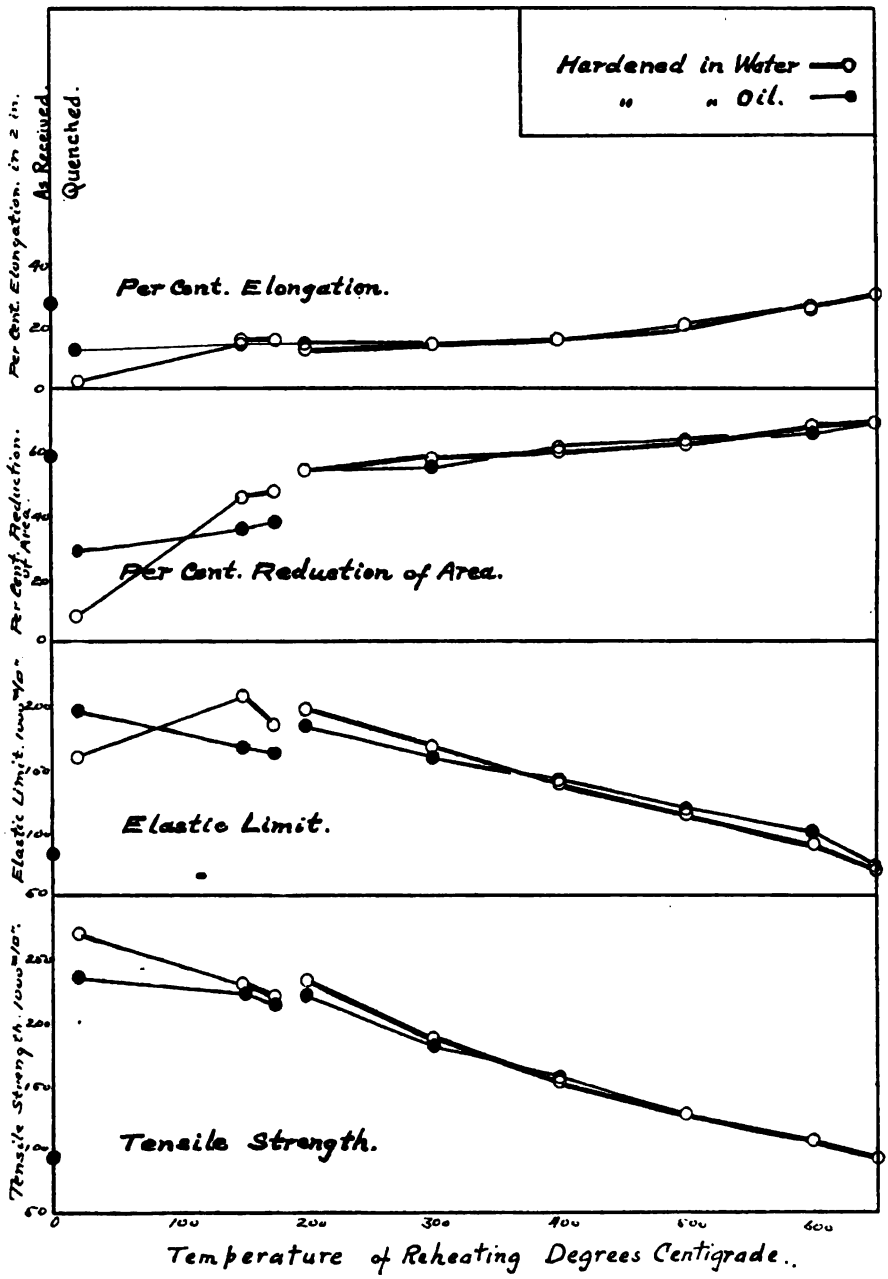
Hardened in Water
" " Oil.
Quenched.
As Received.
Per Cent. Elongation. in 2 in.
Per Cent. Elongation.
Per Cent. Reduction of Area.
Elastic Limit. lbs. per sq. in.
Elastic Limit.
Tensile Strength. lbs. per sq. in.
Tensile Strength.
0
100
200
300
400
500
600
Temperature of Reheating Degrees Centigrade.

FIG. 2.

percent. Reheating to 200° C., while it lowers the strength increases the elongation to 12 percent., and the reduction in area to 54 percent. In order to find whether tempering was effective at much lower temperatures, two more pairs of bars were heated to 900° C., quenched, and drawn to 150° and 175° C. These are bars 17_{23-8} and 17_{23-7} water-quenched and 17_{24-8} and 17_{24-7} oil-quenched. Having thus been quenched in another heat, these bars are not strictly comparable with the other six of the series. However, while maintaining a high tensile strength, the ductility is increased enormously by reheating to 150° C. These results are so remarkable that the work will be repeated with a series tempered between 100° and 200° C.

Oil-quenched and tempered.— Table VI shows the effect of

TABLE NO. VI

BARS HEATED TO 900° C., QUENCHED IN OIL AND REHEATED TO VARIOUS TEMPERATURES

No.	Reheated to — °C.	Yield point, lb. per sq. in.	Ultimate strength, lb. per sq. in.	Elongation in 2 in., per cent.	Reduction in area, percent.
17_{42-2}	0	197,300	236,000	10	29
17_{24-8}	150	167,700	221,500	14	35
17_{24-7}	175	161,500	217,000	15	38
17_{24-6}	200	183,450	221,580	14.5	54.6
17_{24-1}	300	159,630	184,150	14	55.7
17_{24-2}	400	142,900	157,750	16	61.6
17_{24-3}	500	117,070	128,470	20	62.9
17_{24-4}	600	98,820.	106,140	25	67.8
17_{24-5}	650	72,920	94,640	30	68.9
17	as received.	83,190	93,175	27	58.7

quenching in oil and reheating. The results are not so markedly different as in Table V, but from 200° C. the effect of reheating is uniform. At 650° C. we get back almost to the original steel, with a little more ductility. Fig. 2 shows these results graphically, and the uniformity above 200° C. indicates that the effects of tempering vary uniformly with the temperature.

From the above results we may pick the following for comparison with other material:

No.	Heated to 900° C. and quenched	Reheated to —° C.	Yield point, lb. per sq. in.	Ultimate strength, lb. per sq. in.	Elongation in 2 in., per cent.	Reduction in area, percent.
17_{20-6}	in water	200	196,000	233,500	12	54
17_{21-6}	in oil	200	183,450	221,580	14.5	54.5

Microstructure.

The structure of this steel shows the same variation of ferrite and pearlite as normal low-carbon steel. The quenched material, however, is more difficult to etch. The best results on quenched and tempered material were obtained with 2-percent. nitric acid in alcohol, followed by a slight repolish to get rid of the stain.

The overheated material is clearly seen to be refined at 750° C., which is at a much lower temperature than for a corresponding carbon steel.[7]

Quenching at 700° C. gives a fine-grained mixture of ferrite and austenite (martensite), while by quenching at 750° we have nothing but martensite. The higher the temperature above this, the coarser the grain. Tempering, of course, has no effect on the grain size, but effects the austenite (martensite) itself, decomposing it more and more, therefore making it etch more and more deeply. In other words the change yields pseudomorphs after austenite; the higher the temperature of reheating the more extensive the decomposition. Lastly, some recent work on the effect of time of tempering shows that at 300° C. there is a marked difference between tempering for a half hour and for 18 hours. This work is to be continued.

Table VII gives the principal changes in microstructure:

[7] W. Campbell: "Further Notes on the Annealing of Steel." *Proceedings*, American Society for Testing Materials, Vol. X, 1910, p. 193.

TABLE NO. VII

STRUCTURE OF NICKEL STEEL AFTER VARYING TREATMENTS

No.	Treatment.	Microstructure.
16	As received	Fine grained mixture of ferrite and pearlite. Outside shows more ferrite and effect of cold drawing
16.0	Heated to white, cooled in furnace	Coarse grain. Characteristic overheated structure.
16.8	Heated; reheated to 680°C.	Pearlite refined (?) Ferrite unrefined.
16.7	" " " 710	Almost refined. A little coarse ferrite remains
16.9	" " " 720	A trace of coarse ferrite unabsorbed.
16.6	" " " 750	Refined completely.
16.5	" " " 790	Grain slightly coarser than last.
16.4	" " " 810	" " " " "
16.2	" " " 820	Same.
16.1	" " " 850	Grain coarser.
16.3	" " " 885	" " A few large grains.
17.22	" " " 900	Same.
17.10	Heated to 700° and quenched	Fine grains of ferrite in austenite.
17.13	" " 750 " "	Very fine-grained martensite.
17.16	" " 800 " "	Slightly coarser martensite.
17.33	" " 859 " "	" " "
17.19	" " 900 " "	Much coarser martensite.
17.44	" " 1018 " "	Coarser martensite.

CONCLUSIONS.

A steel with 3.15 percent. nickel, 0.27 percent. carbon, and 0.65 percent. manganese, may be refined by heating to 750° C., which is about 75° lower than for an ordinary carbon steel.

Hardening is complete when quenched from this temperature, but higher tensile strength was obtained by quenching around 900° C.

Overheating becomes marked above 1200°.

Tempering of bars quenched in water or oil gives very uniform increase in ductility and decreases in strength, the change varying very uniformly with the temperature of reheating, till at 650° C. the properties are very nearly those of the original steel, having a slightly lower elastic limit and greater reduction in area.

ABSTRACTS OF RECENT MINING DECISIONS.*

LOCATION OF CLAIMS.

LOCATION OF MINING CLAIM — RIGHTS OF SUBSEQUENT LOCATORS.

The law requires the locator of a mining claim to make his location so definite and certain that, from the location notice and stakes or monuments on the ground, the limits and boundaries of his claim may be ascertained, and so definite and certain as to prevent the changing or floating of the claim. And where a discovery is made on a vein and mineral bearing rock, and the notice provides that such claim extends 700 ft. in a northwesterly direction and 800 ft. in a southeasterly direction from such discovery, and the corner stakes on the southeasterly end are so placed as to take in more than 800 ft. of such vein, subsequent locators may legally locate this excess of ground, as the first location is valid only to the exten of 800 ft. southeasterly from the point of discovery.

Flynn Groupe Mining Co. *v.* Murphy, (Idaho) 109 Pacific, 851. June, 1910.

NOTICE OF LOCATION.

Where a mining claim appears to have been located in good faith, it will be deemed sufficient, if by any reasonable construction the language used in the location notice, describing the claim and referring to natural objects and permanent monuments, imparts knowledge of its location to a subsequent locator. Under the Idaho statutes the location notice is not required to describe the exterior boundaries of the claim. Under such a notice no technicalities will be resorted to in order to sustain a re-location of the same ground.

Flynn Groupe Mining Co. *v.* Murphy, (Idaho) 109 Pacific, 852. June, 1910.

FAILURE TO DISCOVER ORE IN MINING DISCOVERY SHAFT.

The failure to discover ore in a discovery shaft sunk on a mining claim does not affect the validity of the claim or the claimant's title, where the locator subsequently found valuable ore in other workings on the claim, if the work done in other parts of the claim, where ore was unquestionable discovered, was more than the equivalent of that required for the discovery shaft.

Gibson *v.* Hjul, (Nevada) 108 Pacific, 759.

EXCESSIVE MINING LOCATION.

An excessive mineral location made through mistake, where the locator was acting in good faith, renders the location void only as to

* Prepared for the SCHOOL OF MINES QUARTERLY by J. W. Thompson, Attorney-at-Law, Indianapolis, Ind.

the excess; but where the locator has purposely included within his exterior boundaries an exclusive area for the fraudulent purpose of holding the entire area then it all becomes void. So if the location is made so large that it cannot be deemed an innocent mistake, fraudulent conduct in the location may be presumed. And where the exterior boundaries include such an unreasonable excessive area that such boundary lines cannot be said to impart notice to a prospector of a mineral location or discovery within a reasonable distance of a lawful claim, then the location will be held void on the ground that the boundaries of the claim have not been marked and established as required by law.

Nicholls *v.* Lewis & Clark Mining Co., (Idaho) 109 Pacific, 846. June, 1910.

Notice of Location of Mining Claim.

The statute of Nevada does not require the notice of the location of a mining claim to be recorded. And a notice of location of such a claim, though not in strict compliance with the statute as a certificate of location, and therefore not evidence of an act of location, is admissible in support of a claim in adverse possession. The rule is that such a notice is not required to be strictly exact, and is not controlling, and that courses and distances given therein yield to the monuments erected on the ground. It is also the rule that the filing of a defective certificate of location, or the failure to file any certificate, does not invalidate the claim. Neither does the fact that the end lines of a mining claim are not parallel invalidate the location but only affects extralateral rights.

Gibson *v.* Hjul, (Colorado) 108 Pacific, 759.

MINING LEASES.

Taxation of Minerals.

The statute of Kansas providing for the taxation of strata of minerals in land, the title to which has been vested in persons other than the owner of the surface, applies to oil and gas as well as to solid minerals; and under this statute where the different strata are severed by a conveyance or otherwise, each stratum is subject to be taxed separately as real property, and it is the duty of the owner not only to record the instrument which conveys the property to him within the time specified, but also to see that it is duly listed for taxation at the proper time. On this principle, an instrument called a lease, by which the owner of the land grants, conveys and warrants to another all the coal, oil and gas under a certain tract of land, operates to sever the coal, oil, and gas from the remainder of the land and the interest segregated and conveyed is subject to be separately taxed, and it was the duty of the owner of such interest to list it for taxation.

Mound City Brick & Gas Co. *v.* Goodspeed Gas & Oil Co. (Kansas) 109 Pacific, 1002.

NATURE OF OIL AND GAS LEASE.

Oil and gas while in the earth are unlike solid minerals in that they are not subject of the ownership distinct from the soil; and accordingly a lease or grant of the oil and gas is in effect a grant not of the oil that is in the ground, but of such a part as the lessee may find, and such a lease passes no estate that can be the subject of an ejectment or other real action.

Kolachny *v.* Galbreath, (Oklahoma) 110 Pacific, 902.

LIABILITY FOR INJURY TO MINER.

LIABILITY OF MINE OWNER FOR INJURY FROM IMPROPER VENTILATION.

The statute of Pennsylvania requires the owner of a mine to place the mine and the underground works in charge and daily supervision of a mine foreman. The statute commits to such foreman the charge of matters pertaining to the ventilation of the mine and prohibits the superintendent from interfering with him in the performance of these duties. Accordingly where the owner of the mine placed a competent mine foreman in charge of the mine, whose duties require him, among other things, to protect the miners against insufficient or inadequate ventilation, such mine owner has complied with the statute requiring him to use precaution for the safety of the miners, and the mine owner is not liable for injury to a miner resulting from neglect or failure of such foreman properly to conduct the air currents along the face of the working places of the mine. In an action to recover damages for injuries to a miner sustained by an explosion of gases in an anthracite coal mine on an alleged failure of the mine owner, a corporation, to perform its duties, it was conceded that the company had employed a competent certified foreman and that at the time of the accident he was in charge of the underground workings of the mine. There was no evidence however that the owner had not provided and maintained an adequate supply of pure air for the mine. Neither did the evidence show that the owner had in any manner failed to perform the statutory duties imposed on it, unless it should have driven the top-heading in order to permit the air currents to circulate in the breast above the place where the miner was injured. The evidence, according to the decision of the court, showed no failure on the part of the owner to furnish any material required by law, the want of which contributed to the accident. By command of the statute the interior of the mine is taken out of the possession and control of the owner and placed in charge of a certified foreman, with whom the owner's superintendent is forbidden to interfere and who has power to compel compliance with his directions so far as they relate to the safety of the miners engaged in the mine. It was therefore the duty of the mine foreman to provide such headings as were necessary to ventilate the mine, and it was his duty, and not that of the owner, to determine the necessity for another heading in the breast and to provide such heading if good ventilation in that part of the mine required it. Anything and everything that effects the

health and safety of the workmen while engaged at their work is by the statute placed in the keeping and charge of the mine foreman. Whether the air current should have been conducted to one breast by means of brattices or a top-heading should have been driven to another breast or a heading and any other location in the mine should have been driven for the purpose of ventilating the breast where the accident occurred, were questions to be determined and matters to be regulated by the mine foreman.

Dempsey *v.* Buck Run Coal Co., (Pennsylvania) 76 Atlantic, 745. March, 1910.

STATUTE MAKING MINE OWNERS LIABLE FOR NEGLIGENCE OF MINE FOREMAN IS UNCONSTITUTIONAL.

The section of the Pennsylvania statute of 1891 providing that for any injury occasioned by violation of the act for failure of the mine owner to comply with its provisions a right of action shall accrue to the party injured against such owner for any damages he may have sustained, is unconstitutional in so far as it imposes a liability upon the owner of the mine for failure of the mine foreman to comply with the statute.

Dempsey *v.* Buck Run Coal Co., (Pennsylvania) 76 Atlantic, 745. March, 1910.

DUTY OF MINE OWNER IN CASE OF NEGLECT OF MINE FOREMAN.

Where the owner of a mine knows that the mine foreman is neglecting the performance of his duties, or that the mine is in a condition which endangers the health or the safety of the miners, it is his duty to act promptly and have such dangers removed; but where the owner himself is not in fault he cannot be made liable for the neglect of duty imposed by the statute upon the mine foreman. He necessarily must operate his mine through others and when he complies with all statutory provisions enacted for the protection of his workmen by employing competent assistance, he has done all that is required of him and cannot be subjected to liability for which he is not responsible.

Dempsey *v.* Buck Run Coal Co., (Pennsylvania) 76 Atlantic, 745. March, 1910.

LIABILITY FOR DEATH OF PERSON RESCUING IMPERILED MINERS.

A complaint for damages for the death of a rescuer, who perished in a mine, alleged that a fire had been burning for some days in a coal mine and during such time one of the ventilating fans had been stopped for one day and by reason thereof gases accumulated in the mine through which men going in to subdue the fire must pass; that on the next day the fan was started in the mine but had not been running a sufficient time to clear the passageways of the gases; that the corporation operating the mine did not examine to ascertain the condition of the passageways, but the superintendent and general manager of the company sent several miners in to subdue the fire with-

out being informed as to the presence of these gases and being overcome with them were in peril of their lives; that at the request of the superintendent, the deceased, without knowledge of the prevailing conditions, entered the mine to aid in the rescue; that the defendants, wholly disregarding their duty, negligently failed to inform the deceased as to the existence of the poisonous gases that had accumulated in the mine, that were existing from the negligent acts and conduct of the defendants, and negligently failed to advise the rescuer, the deceased, of the gases then existing in the mine, and that the said rescuer ignorant of such lack of ventilation and suspecting and believing that the only dangers to which he was then exposing himself in the work of rescue, were the dangers and risks which arose from the gases then being created on account of the fire in the mine, and that he entered the mine for the purpose of rescuing the persons therein, and after so entering and while performing the work of rescue was overcome by the gases in consequence of which he died. The recovery was sought upon the theory that the defendants were chargeable with the death of the rescuer by requesting or permitting him to enter the mine for the purpose of rescuing the imperiled miners without informing him of the dangerous condition known or which should have been known to them to exist therein and thus exposing him to a peril of which he had no knowledge. The alleged peril did not exist by reason of the fire, nor of the gases generated by it, but the peril of the miners was due to the accumulation of gases spontaneously generated in the unused workings, and that the accumulation of such gases in the passageways was the cause of the death. It is a rule of law that the one who, observing another in peril, voluntarily exposes himself to the same danger in order to protect him or save his life may recover for any injury sustained in effecting the rescue against the person through whose negligence the perilous condition was brought about, provided however the rescue is not made under such circumstances as to constitute rashness in the judgment of prudent persons. The rule rests upon the principle that it is commendable to save life, and though a person attempting to save it voluntarily exposes himself to danger, the law will not readily impute to him responsibility for an injury so received. The incurring of the danger in such cases is not, per se, negligence, and the question of contributory negligence depends upon the proof of the circumstances surrounding the attempt to rescue, such as the alarm, excitement, and confusion usually present, the uncertainty as to the means to be used, and the liability to err in the exercise of judgment as to the best course to pursue, and great latitude of judgment is allowed to one who is impelled by the dictates of humanity to decide and act in the face of emergencies. The court however found that the death of the rescuer was caused, as shown by the evidence, from gases generated by the fire, and that, inasmuch as the complaint alleged and the case proceeded on the theory that the rescuer's death was due to the inhalation of gases other than those generated by the fire, and having failed to furnish evidence to establish, directly or indirectly, that the

death was caused as alleged, there could be no recovery, as the divergence thus appearing between the cause of action as alleged and the evidence adduced to establish it, is such a variance that it amounts to a failure of proof, and brings the case within the rule that, unless evidence furnishes substantial proof of the cause of action as alleged, the plaintiff has failed to make out his case, even though the case shows negligence in other respects. The rule is that the evidence must tend not only to show the negligence alleged but also the casual connection between it and the injury.

Bracey *v.* Northwestern Improvement Co., (Montana) 109 Pacific, 706. June, 1910.

Care Required of Operator of Mine.

An operator of a mine must use such care for the protection of the miners as would be exercised by an ordinarily prudent man in the same business, and he must observe the same rules in selecting the tools and appliances furnished the miners. Accordingly, where the operator of the mine selects an appliance in general use and reasonably adapted to the purpose for which it is used, a mere continuance of its use does not of itself indicate negligence, though there may be safer devices used by other operators to accomplish the same purpose. When an appliance is not obviously dangerous, but has been proved by long use to be safe and efficient, its use may be continued. An operator of a mine, like other employers, is not required to furnish the best appliances nor the safest, nor provide the best method for the operation, as such a rule might require a continued change of appliances and tools.

Cummings *v.* Reins Copper Co., (Montana) 107 Pacific, 904.

Injury to Miner — Liability of Operator.

A mine operator was not liable in an action for injuries to a miner injured while hoisting timbers by means of a rope and block with a pulley, where the evidence showed that a similar rope had been used by other miners for a long time with profit, efficiency and safety, and that while it was not so well adapted for the work as it would have been had it been fitted with an iron or steel attachment, it was not obviously dangerous and the mine operator was not negligent in failing to furnish a rope with iron or steel attachment. And in no event would the mine operator be liable where the evidence showed that the timber escaped because the rope was not properly tied by the miner; neither would he be liable where the evidence showed that the timber escaped from some unknown cause. To make the mine owner liable in such a case it is necessary not only to show that the miner was injured during the course of his employment, but that the injury was proximately caused by the negligence of the mine operator.

Cummings *v.* Reins Copper Co., (Montana) 107 Pacific, 904.

Injury to Miner — Safety Cages.

The statute of Montana requires iron-bonneted safety cages with

doors, in mine shafts over 300 ft. deep, to be used in lowering and hoisting the miners. The provision that the doors must be closed "when lowering or hoisting the men," must apply to the singular number as well as the plural, and requires the closing of the doors during the hoisting or lowering of one or more miners. The statute was not complied with by closing the doors only when the shifts of men employed in the mine were being lowered and hoisted at the usual time of changing the miners. And where the statute provides that when the cage is used for sinking only it need not be equipped with such doors, the term "sinking" is not technical but means descending, going down, excavating downward without substantial deviation from a straight line, and does not include the cutting of stations so as to relieve the operators from providing doors to such cages during such operations. Evidence in the trial for an injury to a miner while in such cage, showing that he entered the cage from a station from one side and that his foot got off or over on that side where it was crushed, was sufficient to show that his foot extended out of the cage through an opening where there should have been a door, and made it clear that the absence of the door from the cage was the proximate cause of the injury.

Osterholm *v.* Boston & Montana Consolidated Copper & Silver Mining Co., (Montana) 107 Pacific, 499.

CONTRIBUTORY NEGLIGENCE.

A miner entering a cage for the purpose of ascending the shaft under the direction of the foreman of the mine was not guilty of contributory negligence where, in attempting to ring the bell to signal the hoist, he found the bell rope wet and the bell hard to ring, and in the attempt he slipped, struck his elbow and fainted, and in falling caught his leg between the wall and the cage resulting in the injury complained of.

Osterholm *v.* Boston & Montana Consolidated Copper and Silver Mining Co., (Montana) 107 Pacific, 499.

INSPECTION OF MINE — LIABILITY OF MINE OWNER.

The statute of Illinois requires a mine owner to employ a legally qualified mine examiner to inspect the mine before miners are permitted to enter for work and to see that all dangerous places are properly marked, as required by the statute. This statute is mandatory and not merely discretionary, involving only the exercise of good faith on the part of the mine owner; and a mine owner cannot excuse himself from liability for a failure properly to examine the mine and mark the dangerous places, even where the mine examiner and mine manager examined the place where the injury occurred, and in good faith believed it was not dangerous. And where the rock and other débris, which fell and struck a miner, fell by reason of the condition of the roof of the mine before he commenced work, and whether the place should have been marked as dangerous are questions of fact to be determined by a jury under the evidence.

Aetitus *v.* Spring Valley Coal Co., (Illinois) 92 Northeastern, 579.

EVIDENCE — REPORT OF MINE SUPERINTENDENT.

The statute of Illinois requires a mine superintendent to make a record of any accident occurring in the mine and to make a report of the same to the state mine inspector. Under this statute, such a report, and the record made by the mine superintendent to the mine inspector, are admissible in evidence in the trial of a case for an injury to a miner to establish the facts required to be so reported and recorded; and the fact that such a report contained a statement that a piece of rock fell from the roof of the mine and struck a miner, and that the miner had a wife and child, would not operate to render such report and record inadmissible as evidence.

Aetitus *v.* Spring Valley Coal Co., (Illinois) 92 Northeastern, 579.

LIABILITY OF MINE OWNER FOR DAMAGES TO MINER CAUSED BY THE NEGLIGENCE OF ANOTHER MINER.

The statute of Montana makes a mine operator liable for damages to a miner injured through the negligence of another miner. It also provides that in case of death of the miner from any injury so sustained, the right of action shall survive to and may be prosecuted by his heirs. Under this statute, in an action by the heirs for the death of a miner caused by the negligence of another miner, the scope of the recovery is the same as his would have been but for his death, including damages for the pain and suffering he endured and for his diminished and lost earning capacity for the period of his natural expectancy. And where such an action was based only on negligence of the engineer in handling the cage, evidence was admissible to show that the cage was without doors, as bearing on the degree of care that the engineer with knowledge of such fact was required to exercise in handling the cage.

Beeler *v.* Butte & London Development Co., (Montana) 110 Pacific, 528. July, 1910.

INJURY TO MINER — MEASURE OF DAMAGES.

An employe or miner negligently injured is entitled to damages fairly compensating him for his mental and physical suffering, loss of time, permanent impairment of money earning power proximately resulting from the injury, not exceeding the amount claimed in his petition.

West Kentucky Coal Co. *v.* Davis, (Kentucky) 128 Southwestern, 1074.

DUTY OF MINE OWNER TO KEEP MINE SAFE.

In an action by a miner injured by the falling of rock from the roof of an entry to a mine, he cannot recover without alleging and proving actual or constructive notice to the mine owner of the danger, where such an action is based on the master's common law duty to keep a safe place for his servants to work in. But where such an action is based on the duty imposed by the statute of Kansas requir-

ing a mine owner to keep careful watch to see that rock overhead is carefully secured against falling as excavations are advanced, a miner injured by rock falling from the roof can recover without alleging and proving that the owner had actual notice that the rock was loose, or that the rock was in such condition long enough to charge the owner with constructive notice. Under the statute, a miner has the right to rely on the performance by the mine owner of the duties imposed upon him by the statute to inspect and keep the roof of the entry securely propped so as to prevent such an accident, and the miner is not guilty of contributory negligence for failure to see that the rock was loose. This statute was not intended merely to declare the common law duty, but it requires more than ordinary diligence to furnish a safe place for the miners to work. And the negligence of the foreman to perform these statutory duties imposed upon the master is the negligence of the master himself.

Little *v.* Norton Coal Co., (Kansas) 19 Pacific, 768. July, 1910.

DEATH OF MINER BY SUFFOCATION — ASSUMPTION OF RISK.

In an action for damages for the death of a miner caused by suffocation it is proper to prove that the superintendent directed the miner to enter the mine and told him that it was perfectly safe. And such evidence is competent in determining the question of the assumption of risk and of contributory negligence on the part of the miner. And where the miner was informed by a third person in the presence of the foreman that it was safe to go into the mine and the foreman immediately directed the miner to go ahead, that everything was all right, the court cannot say as a matter of law there was existing a degree of danger calculated to deter a person of ordinary prudence from the undertaking, and such as to make him guilty of contributory negligence in going into the mine.

Alabama Consolidated Coal & Iron Co. *v.* Heald, (Alabama) 53 Southern, 162.

INJURY TO MINER BY FALLING ROCK — LIABILITY OF MINE OPERATOR.

In an action for the death of a miner who was killed by falling rock from the roof of a tunnel, it was held competent to show that rock and earth had fallen from the roof at different times prior to the accident and at other places for the purpose of showing the character of the ground and the necessity of supporting the roof, and as a means of conveying notice to the mine owner and operator to the defective and dangerous condition of the roof.

Sargent *v.* Union Fuel Co., (Utah) 108 Pacific, 928.

ASSUMPTION OF RISK.

A person entering voluntarily into a contract of service assumes all the risks and hazards ordinarily incident to the employment and such as are liable to arise from defects which are patent and obvious to a person of his experience and understanding, and he assumes all the ordinary risk of the employment which are known to him and

which would have been known with the exercise of ordinary care, it is his duty to exercise care and diligence and observe and become cognizant of obvious defects in the machinery and the work place, and he is charged with the knowledge of such defects which would have been known to a person of reasonable care and prudence. But notwithstanding this rule, an employe does not ordinarily assume risks arising out of the negligence of his employer. This rule was applied in an action against a mine company for the death of a miner, acting in the capacity of a shot firer, resulting from injuries caused by smoke, fumes, foul air and poisonous gases by a reversal of a fan on account of its not having been suitably adjusted with reasonably necessary appliances for the purpose for which it was installed, and where it was shown that the omitted appliances were reasonably necessary in its operation for such purposes. The omission of such suitably adjusted fan with such additional appliances was held to be the proximate cause of the injury.

Colgate Co. *v.* Hurst, (Oklahoma) 107 Pacific, 657.

COAL LANDS.

Deed for Coal and Mining Privileges — Right to Subjacent Support.

Where a deed conveys the coal under a tract of land, together with the right to enter upon and under the land, and to mine, excavate, and remove all the coal under such land, there is no implied reservation in such a deed that the grantee shall leave coal enough to support the surface in its original position, but on the contrary is a waiver or surrender of such right, and the grantor cannot recover damages for injury to the surface resulting from the removal of all the coal without leaving sufficient support.

Kuhn *v.* Fairmont Coal Co. 179 Federal, 191.

Combination to Procure Title to Coal Lands.

The fact that two persons engaged in an unlawful combination to procure title on behalf of a corporation to an area of coal lands in excess of the amount prescribed by law, and that lands in excess of the amount allowed by law were actually patented to them, would not make such patent valid, the unlawful combination making the proceeding illegal from the beginning. And a corporation formed to take over such patent lands is not a bona fide purchaser for value without notice to the extent that it precludes the government from proceeding to cancel the patent, as the rule is that the person holding a voidable patent to public lands cannot protect himself against the process of the government by forming a corporation in which he is the dominant factor and conveying to it the premises which he has acquired in violation of law.

United States *v.* Allen, 180 Federal, 855.

MISCELLANEOUS.

Mine Appliances or Fixtures — Sale on Execution.

Engines, boilers, mills, pumps and electric hoist firmly bolted to the substructure on which it rests and the superstructure and engine house surrounding it, sufficiently affixed to the soil for mining purposes, and a gallows frame at the mine, the base of which was originally sunk into the earth to make it level and substantial for use for mining, together with the gallows, hoist, and transformers forming integral parts of one mechanism essential to the operation of the mine, are fixtures and are not subject to sale on execution as personal property under the laws of Nevada. This rule applies where a buyer of chattels under a contract stipulating that the same shall be regarded as the personal property of the seller, with the right of removal until paid for, and thereupon attaches such property to real estate so as to make the same fixtures, they do become fixtures as against every other person except the seller, and cannot be sold as personal property on the execution against the buyer. A sale of property as personal property at an execution sale is invalid where any of the articles sold were fixtures.

Arnold *v.* Goldfield Third Chance Mining Co., (Nevada) 109 Pacific, 718. July 1, 1910.

Life Insurance — Visiting Mine Not a Forfeiture.

A condition in a benefit insurance certificate containing a prohibition against the insured engaging in prospecting or mining is not violated and the insurance is not forfeited by merely visiting a mine.

Butler *v.* Supreme Court, etc., Independent Order of Foresters, (Washington) 110 Pacific, 1007.

Contract for Purchase of Mining Claim.

A contract for the sale of mining claims and stipulating that such claims and all machinery placed thereon in working the mine shall be the property of the vendor until the purchase price is paid, and providing that the purchaser shall work and develop the mine and extract ores and appropriate the proceeds, gives such purchaser the right to prospect and develop the mines, but he cannot commit waste or remove from the place any property constituting a part of the realty other than the ore so mined. In such case the machinery on the claim for use in working the mines becomes fixtures attached to the mine, and such machinery cannot be attached by the creditor of the purchaser while the purchase price remains unpaid.

Conde *v.* Sweeney, (California) 110 Pacific, 973.

Option on Mining Claims.

A contract to convey mining claims for a specified sum in cash and other payments to be made in one or two years, with a stipulation that the purchaser might take possession of the claims and develop them, and providing that the purchaser should abide by the terms of

the agreement and in default of any of the payments to surrender such claims to the owner and forfeit all moneys paid and expended, is nothing more than an option, and the purchaser is left to his election within the life of the agreement to purchase or a forfeiture of all sums paid or expended. But the purchaser entering under such an option and employing laborers to perform the work, can establish no indebtedness in favor of the miners against the owners, and no lien attaches to his interest in the property, as such work is not performed by virtue of any employment from the owner, and the person holding such option is neither a vendee nor an agent of the owner. In such an option time is said to be of the essence of the contract and a failure of the original purchaser or his assignee to pay the balance due on the specified dates relieves the original owner of any obligation to convey under the option agreement and all rights are forfeited.

Harper *v.* Independence Development Co., (Utah) 108 Pacific, 701.

PROPERTY SUBJECT TO MINERS' LIENS.

The statute of Oregon provides that when two or more mines are owned by the same person or corporation and worked through a common shaft or tunnel, or at one mill, or other reduction works, then all such mines, and all roads, tramways, trails, flumes, ditches, or pipe lines, buildings, structures or superstructures used or owned in connection therewith, shall be taken and deemed as one mine. And in an action to enforce miners' liens the words "roads, tramways, flumes, ditches or pipe lines," were held to include such appurtenances though not situated upon the mine, and that the term "upon any millsite or mill used, owned, or operated in connection with such mine," had reference to millsites or mills not situated on the mining property, so that a miner's lien notice need not state and it is not necessary to prove that the labor for which the lien is claimed was done on the mill or building on the mine.

Washburn *v.* Inter-mountain Co., (Oregon) 109 Pacific, 382.

TAXATION OF MINING CLAIMS — DESCRIPTION OF CLAIM.

The statute of Arizona provides that the assessor in preparing an assessment roll shall describe real estate by common designation or name, and it also provides that the description shall be sufficient if the land or premises can be readily identified thereby. Under this statute the supreme court of Arizona held that the description of a patented mining claim was sufficient as the name of such claim was a "common designation or name" within the meaning of the statute, and that by such name the premises might be "readily identified." The court also suggested that the practice of assessing patented mines by name has obtained too long in the territory to be questioned, and that such practice is a practical construction of statute as the mode of assessment, and of which the courts may take judicial notice.

Territory *v.* Copper Queen Consolidated Mining Co., (Arizona) 108 Pacific, 960.

MECHANICS' LIEN ON MINING PROPERTY — FILING.

Where the mining property and all the improvements are in the hands of a receiver, and all the parties in interest are brought into court within four months after the date upon which material was last furnished or labor last performed under a contract, this will be deemed a sufficient filing of the claim within the Oklahoma statutes and the court may adjust the rights of all lien-holders. And a statute authorizing mechanics' liens on improvements applies to a coal mine, as a coal mine is an improvement within the meaning of the law.

Peaceable Creek Coal Co. *v.* Jackson, (Oklahoma) 108 Pacific, 409.

SUMMER ACTIVITIES.

CAMP COLUMBIA.

The summer school of surveying opened for the summer of 1911 on June 2 and the last student left on September 16, just in time to take part in the fall deficiency examinations. The summer was, on the whole, a successful one and resulted not only in the accomplishment of the required work but also in many pleasant recollections.

Camp Columbia has, of course, changed immensely during the last five or ten years. In a recent article in the *Alumni News,* the writer has tried to describe these changes and to outline what is being done at Camp to-day. The great increase in the number of students in the University has meant a correspondingly great increase in the number attending Camp, for the present arrangement requires all first-year students in the School of Mines, Engineering and Chemistry to go to Camp for at least one course. Where in former years an attendance of 100 was considered very high, it is now considered rather low and in 1910 a maximum attendance of 181 was reached. This last summer the attendance was much lower; apparently a large number of men were either financially unable to come to Camp or were so well fixed that they could afford to pay the extra fees and do their Camp work all in one summer later on. The maximum attendance was 133 and the minimum 79 with 110 as an average.

It has been customary to give some statements in this report regarding weather conditions, so not to neglect any former customs we may state that Camp opened with three days of cold rain, then the weather man gave us an exceptionally good "surveying summer," although Mr. Skilton said it was "durn hard on crops." During the last two weeks, crops got their innings again and we had those delightful cold, rainy days when our only consolation was that we were not out in tents, as in the olden days, and that there was a fire-place in the Y. M. C. A. building. September 14 brought a heavy frost, with ice fully one-quarter of an inch thick on the watering trough at the barn.

The principal events of the summer were Crew Night, President's Day, Baseball Game and Dinner, and the night we got final clearance. The first two of these were new things and it looks as if they would certainly have to become fixtures. The day of the Poughkeepsie races was spent by the Freshmen in piling up a lot of brush for a bonfire in the evening. Later in the afternoon everybody turned in and helped. We were prepared to celebrate a victory on the Hudson and although we did feel rather glum over the outcome we had a celebration in honor of the great race the Varsity rowed and the Freshman victory. A few kegs of beer, and some songs, made the first Crew Night a successful substitute for the annual "keg party."

President's Day was followed by a general swelling of heads all around Camp. Even the chef felt elated because the President had told him it was a good dinner and a good kitchen, all the students felt rather chesty because Prexy said they were doing good work. On July 3 the thermometer reached 100° but President and Mrs. Butler, Mr. B. B. Lawrence, and Prof. Perry seemed to forget all about the heat and thoroughly inspected the buildings, the maps and reports and all the other details of the Camp work. After dinner at "Rector's" we listened to a few words from the President, who then introduced our oldest alumni trustee, B. B. Lawrence, who in turn left Professor Perry to tell us how much we resembled the Greeks and how much we were indebted to them for names for our surveys and instruments. We all appreciated Dr. Butler's taking that hard and tiresome trip to inspect Camp and hope that we may have an annual President's Day and be able to welcome the same party to Camp Columbia next summer.

Among other visitors were W. F. Turnbull '03, G. Denison '04, R. B. Macbeth '06, H. Perrine '08, G. J. Lyon '04, O. K. Hand '86, Messrs. E. H. Frost and Riley of the *Engineering News*, and W. L. E. Keuffel of Keuffel & Esser. We missed our annual visit from Dean Goetze, who was in Europe most of the summer. Professor Burr spent August 5 at Camp and was accompanied by Mrs. Burr and Miss Burr. A special luncheon was served, at which Prof. and Mrs. Jacoby and Dr. and Mrs. Mitchell were also present.

In regard to the work itself we also had a very satisfactory summer. The pacing survey was under the direction of E. H. Dickenson '11 and W. M. Grant '12. The farm survey and differential levelling were in charge of Prof. W. S. Williams, of Missouri University, and Chief Engineer of the Pickle Hill Railroad. Prof. Williams came to Camp to see how we conducted our courses in surveying, and left with us not only many valuable points gained from his long experience in Government work, but also fragments of his humor, and stories, which, combined with his good fellowship and readiness to help us all in our work at any time, made the parting with "Prof. Bill" a rather hard one. A. D. Duffie '13 took up the work Prof. Williams had in hand when he left on September 1. The azimuth traverse and city surveys were in charge of A. H. Patterson, '11, and M. F. Sayre '07 looked out for the repetition and mine claim surveys.

R. W. Briggs '11 was the social representative of Camp and had charge of the topographic and plane table surveys, and also the hydrographic work. The profile levelling, as well as supervision of the instruments and supplies, was in the hands of W. D. Tupper '11, assisted in the instrument house by J. H. Fisher.

The geodetic work was in charge of Professors Jacoby and Mitchell, assisted by W. M. Smith, of Lafayette College, and "Deacon" Nosworthy, '07. Each man was required this year for the first time to make latitude determination with the zenith telescope.

The numbers of men taking the various courses were as follows:

Freshman course—Whole course, 81. Part of course, 89.

Sophomore course for C. E's—Whole course, 28. Part of course, 17.

Sophomore course for E. M's—Whole course, 35. Part of course, 27.

Hydrographic course and geodesy—22.

Railroad course for E. M's.—42.

Railroad course for C. E's.—26.

In closing this brief summary of Camp activities for 1911, we would like to urge the officers of the University and Alumni to spend at least a week-end at Camp Columbia next summer, inspect the Camp, renew old acquaintances or make new ones, and spend a few days at the summer resort of the Columbia Spirit.

J. K. FINCH.

DEPARTMENT OF GEOLOGY.

The field instruction of the students in the School of Mines and of the gradute students was given during the first week in June at Middletown, N. Y. The map of the U. S. Geological Survey covering the Goshen Quadrangle was traversed by the class and will furnish the basis of the students' reports. The party consisted of 52 students and five officers, and passed a very instructive week.

Later in the summer, Professor Kemp undertook a study of the mineral springs at Saratoga to form a chapter in the forthcoming bulletin of the New York State Survey which will deal with the local geology. The geology has been prepared by Professor H. P. Cushing of Adelbert College, Cleveland, and Dr. R. Ruedemann of the State Geologist's office.

Professor Grabau gave courses in the summer session at the University, and afterward conducted a party of 12 or 15 students for two or three weeks' work upon the strata of New York State. The party went up the Hudson, turned west at Albany and terminated the trip at Buffalo and Niagara Falls.

Professor Berkey continued his work upon the Tarrytown and West Point sheets for the New York State Survey. He also did much work for the New York City Board of Water Supply. His bulletin on the "Geology of the New Catskill Aqueduct" has been issued by the State Geologist and makes a volume of nearly 300 pages, with many plates and cross-sections. It was written especially to bring out the applications of geology to an engineering problem of the first magnitude. As such, it has been received by engineers all over the country and the volume has been in great demand. Professor Berkey also made a month's trip to Idaho during the summer, revisiting a district which is being studied by W. L. Barrows as a subject for his Ph.D. dissertation.

Several changes have taken place in the staff of the Department. J. E. Hyde, for several years Assistant in Palæontology, leaves us to become Assistant Professor in the Department of Geology at Queens University, Kingston, Ontario. D. D. Condit, Assistant in Geology, takes up his work with the Ohio State Geological Survey. Dr. Elvira

Wood, Curator in Palæontology, has been appointed to a similar position in the Harvard Museum, and has been succeeded by Dr. F. F. Hahn, who comes to us from the University of Munich. Dr. Hahn has entered vigorously into the work and many interests of the Department.

Four gradute students of previous years received the Ph.D. degree in Geology at the last Commencement. C. E. Gordon is Professor of Geology at the Massachusetts Agricultural College. Edwin Kirk is in the United States National Museum as Palæontologist. F. C. Lincoln becomes Assistant Professor of Mining in the University of Illinois. G. S. Rogers was appointed upon the U. S. Geological Survey, and has spent the summer mapping the coal seams along the borders of North Dakota and Montana. Dr. Rogers was placed almost immediately in charge of a field party.

Several interesting appointments have been received by students of the Department who have been away from us for a year or two. Charles S. Stewart, for three years Instructor at Cornell University, becomes Professor of Geology in the State University of Idaho, at Moscow. Victor Ziegler, M.A., 1910, who was the past year Instructor in the State College of Pennsylvania, has been called to an assistant professorship in the South Dakota School of Mines, at Rapid City. T. C. Brown, Ph.D., 1910, who has been teaching at Middlebury College, Vermont, for the last two years, becomes Assistant Professor of Palæontology in the Pennsylvania State College.

The Department opens the year with an unusual number of graduate students, not only from the United States, but from one or two foreign countries. J. F. K.

MINING TRIPS.

Last summer 40 students were engaged in mining field work (Mining 58 and 59). About the middle of June the entire class were given one week's practice in underground surveying in a large colliery at Hazleton, Pa., courteously put at our disposal by the Lehigh Valley Coal Company. Mr. Judd, with two assistants, preceded the class by about three weeks for the purpose of making a preliminary survey of the portion of the mine to be surveyed by the students. He was afterwards joined by Professor Kurtz and four more temporary assistants. With this force of eight instructors, the work could be prosecuted night and day in three shifts of eight hours each. This plan has been followed for several years past and works well. It saves time and makes it possible with large classes to carry on the surveying with fewer instruments.

On completion of the surveying, the students, in squads or individually, began the usual mining summer work, consisting of six weeks of note-taking and sketching. This was done at a number of mines at which the necessary privileges had been previously secured. The assignments of squads are noted below:

Homestake, Mogul, and other gold mines of the Black Hills, S. D. —H. F. Cornwall, G. S. Kearney, H. C. Moffett and G. A. Shaw.

The Pennsylvania and neighboring mines of the Anaconda Copper Co., Butte, Mont.—H. S. Mudd, R. A. Semple, and B. C. Staiger.

Wolftone mine, Tucson, and Moyer shafts and Yak mine, Leadville, Colo.—Two squads, comprising P. Kraft, Jr., B. Trillich, J. E. Bell, S. A. McKeown, J. J. Moore, and J. P. Wood.

Moark and Nemo Zinc Mines, Joplin, Mo.—T. E. Brown, 3d, K. H. Donaldson, E. Hinman, Jr., and G. G. McLochlin.

Burra-Burra mine of Tennessee Copper Co., Ducktown, Tenn.—J. B. Beadle and R. M. Ruthardt.

Victoria Copper Mine, Michigan—J. C. Beam and W. H. McArdle.

Mines of Federal Lead Co., Flat River, Mo.—M. W. Heller and E. N. Kleinbaum.

Copper Queen Mine, Bisbee, Ariz., and Cananea Copper mine, Mexico—P. F. Becker, and R. Tarantous.

Champion, Trimountain and Baltic mines, of the Copper Range Mining Co., Mich.—G. D. Dub, W. A. Scheuch, T. W. Skinner, and M. D. Swartz.

Mines of St. Lawrence Pyrites Co., Stellaville, N. Y.—E. Marquardt, W. M. Norris, Jr., N. R. Sethna, and I. P. Tashof.

Besides these, C. P. Browning, C. T. Carney, J. L. Claghorn L. F. Hazzard, J. M. Lovejoy and I. P. Silverstein, made individual visits respectively to copper mines in Arizona, collieries in Iowa and Pennsylvania, gold mines in Colorado, and an iron mine in New Jersey. Hazzard and Silverstein had received part credit for work previously done in the field, and E. H. Cook was given credit for the entire course.

The results of the summer's field study are in general very satisfactory. A few remarks may be made as to the objects in view. The most immediate and obvious object is to enable the students to gain a first-hand knowledge of the practice of mining, but hardly less important is the incidental training of the powers of observation, together with the cultivation of facility in taking notes and making engineering sketches. It is often necessary to do this work rapidly and under difficulties arising from wet, dirt, and poor light. Notwithstanding reiterated instructions and cautions to the contrary, many students find it hard to resist the temptation to make rough notes and sketches while actually in the mine (perhaps on loose pieces of paper) and copy them afterward in a clean book. This is not only a waste of time but is likely to cause errors in transcribing, and moreover the hasty record or sketch may be found incomplete and inaccurate. The fact that note-books used underground may become soiled and lack mere neatness is relatively unimportant. A stained notebook is just as useful as a clean one. Unavoidable smudges and other marks of underground toil may afterward bring to mind satisfactory recollections of hard and perhaps disagreeable work faithfully done under trying conditions. The author of the immaculate series of note-books who has cast to the four winds his original notes, not only misses this substantial satisfaction, but in after years may have reason to doubt the accuracy of his records. Every student should

remember first that he is doing this work for himself, and second, that good notes form part of his stock-in-trade for the future.

Many members of this year's class merit special commendation for their excellent records of the fundamental mining operations of drilling and blasting. By tabulating repeated observations of the drilling of series of holes, accompanied by notes as to delays and their causes, they have learned much about actual rates of work under different conditions in drifting and stoping. A smaller number of students have done well also in recording the speeds of work in tramming or other underground transport of ore and in hoisting. Space does not permit of any detailed review of the work of the class, otherwise attention might be called to some of the more obvious shortcomings exhibited in the different sets of note-books. Such deficiencies are generally due to thoughtlessness, sometimes to lack of appreciation of present opportunities.

In conclusion the writer would urge upon all students the wisdom of utilizing the valuable opportunities afforded by these summer trips to the mines. The six weeks' time required is short enough and ought not to be clipped too closely. Moreover, the officers of the mines are quick to see and appreciate earnest effort and to lend assistance, and more than once in past years a good position to take effect immediately after graduation has been secured by the intelligent, hard-working student.

The thanks of the School of Mines are due to all the superintendents and other officers of the properties visited by our students for the privileges and assistance so generously given. We wish to express especial appreciation of the kindness of Messrs. F. S. Cook, consulting engineer, Joplin, Mo.; F. T. Rubidge, Vice-President St. Lawrence Pyrites Co., Stellaville, N. Y.; R. S. Schultz, Jr., Engineer of the Victoria Copper Mining Co., Victoria, Mich.; F. W. Denton, Manager of the Copper Range Consolidated Company, Painesdale, Mich., and G. O. Argall, Manager of the Iron-Silver Mining Co., Leadville, Colo.

R. P.

PERSONAL ITEMS.

Walter Harvey Weed, '83, has been appointed consulting engineer in active charge of operations of the Interstate Silver-Lead Mining Company, at Wallace, Idaho.

Clyde T. Griswold, '05, formerly Professor of Mining at Colorado College, Colorado Springs, has been appointed Assistant Professor of Mining Engineering at the Carnegie Technical School, Pittsburg, Pa.

A. P. Rogers, '98, has returned to New York from London, after having spent the last eight months in professional work in Brazil.

Milton H. Fies, '04, has resigned his position as mine superintendent with the Birmingham Coal and Iron Company, to become superintendent of the Banner mines of the Pratt Consolidated Coal Company, Birmingham, Ala. Mr. Fies was one of those appointed by Governor O'Neal, of Alabama, as delegate to the American Mining Congress, meeting at Chicago in October.

George S. Rice, '87, and J. Parke Channing, '83, were appointed by President Taft delegates at large to the American Mining Congress, meeting at Chicago, in October.

George S. Rice, '87, has been appointed a member of a committee of three experts to investigate the supposed undermining of Turtle hill, near Frank, Alberta.

Professor W. Campbell, as chairman of the committee on non-ferrous metals and alloys, of the American Society of Testing Materials, presented a report at the summer meeting at Atlantic City, covering specifications for standard crude copper, spelter and manganese bronze.

J. F. McClelland, 1900, Professor of Mining at Sheffield Scientific School, spent the past summer in the investigation of gold placer deposits in northwest Alaska.

The following graduates have recently been elected to membership in the Mining and Metallurgical Society of America: William A. Wilson, '82, consulting mining enginer, Salt Lake City; Thomas T. Read, '02, associate editor of the *Mining and Scientific Press*, San Francisco, Cal.; and S. F. Shaw, '03, 126 Liberty St., New York City.

William A. Smith, '98, until recently with the United Zinc and Chemical Company, Iola, Kans., has been appointed superintendent of the St. Joseph Lead Company's smelting works at Herculaneum, Mo.

C. N. Nelson, 1905, has left San Javier, Sonora, Mex., to take a position with the Honduras Rosario Mining Company; he left New York on Oct. 18, for San Juancito, Honduras.

Among recent alumni visitors at the School of Mines have been the following: J. F. McClelland, 1900, Professor of Mining at Sheffield Scientific School, New Haven, Conn.; S. J. Kidder, 1904, mill manager of the Silver Peak company at Blair, Nev.; F. H. Seifert, 1906, in charge of the parting department of the Nichols Copper Company's

plant at Laurel Hill, N. Y.; F. H. Clark, 1907, who has now gone on an extended trip of investigation in the West; P. E. Locke, 1909, with the Ray Consolidated Copper Co., at Kelvin, Ariz.; P. VonSaltza, who has left his position on the engineering staff of the Copper Queen, at Bisbee, Ariz., and is now testing gravel deposits on Long Island; J. M. Hoffman, 1910, who is active in the development of Porcupine, Ont.; D. M. Obler, 1910, lately returned from Guanacevi. Durango, Mex., and V. Mulholland, 1911, who contemplates going to take charge of gold dredging works on the Magdalena river, Columbia, S, A.

H. B. Taylor, 1906, and T. C. Morgan, 1911, are assisting Professor Munroe in certain experimental work now being conducted in the ore-dressing laboratory of the School of Mines.

Leicester Durham, C. E., 1899, was married to Viola Ethel Hyde, at Peekskill, N. Y., on October 11, 1911.

Clark G. Mitchell, E. M., 1911, was married on September 21, to Ida Hermine Quentin, at Denver, Colo. Their home will be at Britannia Beach, British Columbia.

Columbia School of Mines is strongly represented on the engineering staff of the Braden copper mine, in Chile. Robert Marsh, Jr., 1907, is superintendent; Horace R. Graham, 1908, and Lester E. Grant, 1909, are shift-bosses; and O. M. Lewyn, 1908, is engineer.

E. H. Dickenson, 1911, has secured a position with the Ely Consolidated company, at Ely, Nev.

William T. MacDonald, 1908, recently with the Utah Copper Company, has been appointed superintendent of the Chino Copper Company's new mill at Hurley, N. M.

A. E. Hall, 1911, is with the North Dome company, at Porcupine, Ontario.

F. C. Lincoln, E. M., Ph.D., 1911, has been appointed Assistant Professor of Mining at the University of Illinois.

Victor Ziegler, A. M., 1910, who was last year Instructor in Geology at Pennsylvania State College, has been appointed Assistant Professor of Mineralogy and Petrography in the State School of Mines at Rapid City, S. D.

G. S. Rogers, Ph.D., 1911, was appointed to the U. S. Geological Survey, and during the last summer has been in charge of a party mapping the coal seams of eastern Montana.

Mason T. Adams, 1901, recently of Denver, has opened an office as consulting engineer at 45 Broadway, New York City.

Sidney Rolle, 1905, is in the smeltery of the United States Metals Company, at Chrome, N. J.

At the annual meeting of the American Society for Testing Materials held at Atlantic City, N. J., in June, the President, Professor Henry M. Howe of Columbia University, read a scholarly presidential address on the advance of the art of testing. Professor Wm. Campbell, of Columbia University, as chairman of the committee "on Non-Ferrous Metals and Alloys," reported three specifications dealing with cast raw materials, and said that its work on wrought metal

is progressing well. However, it has found the field assigned to it too large for unified treatment, and recommends that the work be made more specific by fixing definite subjects and questions to be worked on by the committee.

Copper.— The specifications for copper wire bars, cakes, slabs, billets, ingots, and ingot bars call for a metal of at least 99.880 percent. purity as determined by electrolytic assay (silver being counted as copper). The standard of conductivity (100 percent.) is to be 0.15302 international ohms, which is taken to be the resistance of a metergram of annealed copper at 20° C. Referred to this standard, wire bars and other copper for electrical use are to have at least 98.5 percent. conductivity, and ingot copper 97.5 percent., both in annealed condition; arsenical copper is to have at least 90 percent. conductivity. The specifications fix methods of marking, sampling for tests, etc.

Spelter.— Four grades of spelter are established by the proposed specifications. The maximum impurities allowed are:

Grade.	Lead.	Iron.	Cadmium.	Total not over
A. High grade*.....	0.07	0.03	0.05	0.10
B. Intermediate*	0.20	0.03	0.50	0.50
C. Brass special*.....	0.75	0.04	0.75	1.20
D. Prime Western...	1.50	0.08	.....	.....

* To be free from aluminum.

Methods of sampling, analysis, and settlement of claims are prescribed.

Mangancse-Bronze Ingots.— The third specification of this committee is for manganese-bronze ingots. The chemical composition is prescribed as:

Copper	55 to 60%
Zinc	39 to 45%
Iron, not over	2%
Tin, not over	2%
Aluminum, not over	0.5%
Manganese, not over	0.5%

The standard ½ × 2-in. turned test-piece is to show at least 70,000 lb. per sq. in. ultimate tensile strength, and at least 20 percent. elongation.

There was some difference in the committee on this specification, partly on the score of the name, "brass" being thought more correct than "bronze."

BOOK REVIEWS.

Rock Minerals: Their Chemical and Physical Characters and Their Determination in Thin Sections. Second edition, revised and enlarged. By JOSEPH P. IDDINGS. New York, John Wiley & Sons, 1911. 8vo. xiii+617 pages, 500 figures and one colored chart. Cloth, $5.00.

The first edition was reviewed in great detail in the SCHOOL OF MINES QUARTERLY, vol. xxviii, p. 539, 1907.

The second edition of this valuable book has been bought up to date by the added description of 73 mineral species, including many of the opaque minerals. The general introductory chapters, dealing with chemical, physical and optical characters, have not been changed, with the exception of the brief mention of pleochroic halos (p. 188), resulting from the activity of radiations emanating from uranium, thorium, etc., contained in the mineral inclusions. A new table has been added (in the feldspar group), giving the extinction angles and percentages of anorthite in the lime-soda feldspars.

The free use of paragraphing and the same order always followed in the description of characters, make the looking up of special facts easy and convenient. The heading, "Resemblances," enables the reader to see at a glance the important distinctions between similarly appearing minerals. Students, who do not read German easily, or those who prefer a concise, logical statement of facts, will find this book, which in the introductory portion is largely a translation of Groth and other German authors, a most useful aid in petrographical or mineralogical studies. L. M. L.

Electric Traction and Transmission Engineering. By S. SHELDON and E. HAUSMAN. New York, D. Van Nostrand Co., 1911. $2.50.

About a year ago it was very difficult to find a book on the subject of electric traction which was up to date and suited to the use of students in technical schools, but within the last two months three books dealing specifically with the subject of electric traction have been published.

The volume under consideration is more comprehensive than its title would imply, since it deals not only with the problems of electric traction but investigates at considerable depth the problems of generation and transmission, and even attacks those higher and more recent problems in transmission engineering requiring the use of hyperbolic functions. The first part of the book, dealing particularly with the subject of electric traction, is written in a simple and clear language, which will appeal both to students and engineers of other than the electrical profession, and attacks the problems in a logical and interesting manner. The discussions in the second part on the calcu-

lation of line copper, proportioning of fixed and operating charges, and line transmission problems, involve considerable mathematics and will probably be too deep and too special for the above mentioned classes.

In general the book can be recommended for students in technical schools and will be useful not only during the course but will be referred to after the student has taken up active work. It has been adopted as a text-book at Columbia. In style and size the book is uniform with the previous works by the same authors and has numerous good illustrations and diagrams. W. I. SLICHTER.

Direct and Alternating Current Manual. By FREDERICK BEDELL. New York, D. Van Nostrand Co. 8vo., 360 pp., illustrated. $2.00.

In writing a manual, to be used as a laboratory text, the author has two general methods of treatment open to him. Specific instructions for carrying out the different tests, connection diagrams, prepared blank forms on which data are to be kept, etc., may be given and the analysis of the behaviour of the machine to be tested may be omitted; this method has been employed by several writers in the past. A manual of this type has some advantages; it makes the task of conducting the laboratory work comparatively easy, both for the student and instructor, but it does not tend to make the student think and analyze. The second possible method of treating this subject consists in analyzing, as completely as possible, the principles involved in the construction and behavior of a machine; the method for verifying experimentally the conclusions reached theoretically being left to the student's ingenuity.

The laboratory manual edited by Dr. Bedell follows the latter method. The second edition, recently published, is very much enlarged and covers more thoroughly the field of direct and alternating current testing than did the first edition. An analysis of the more important types of A. C. and D. C. machinery is given, followed by short instructions for performing some of the standard tests; in the appendices, at the end of each experiment, are suggested various other tests which may serve to supplement the standard tests and bring out some of the special features of the machine under investigation.

Some of the analyses are very thorough and complete, while others are not; it seems to the writer that the subject matter is not well enough balanced to fit the book for laboratory use. As an example of this, we note that the treatment of the whole field of direct-current machinery occupies considerably less space than the subject of the static transformer, one of the simplest pieces of alternating-current apparatus. Again the treatment of the synchronous motor and rotary converter is contained in a space about one-third as long as that devoted to the transformer. The analysis of the conditions affecting the parallel operation of machines, both A. C. and D. C., is omitted.

At the end of the book is a chapter devoted to the analysis of complex waves; the treatment is very clear and is the best that the writer has yet seen. Using some of Runge's methods, the author has

reduced the labor of calculating the magnitudes and phases of the upper harmonics to a minimum.

While we do not think the book suitable for a laboratory text, as a reference book on the various topics treated, it will be very useful; the author's ideas on the object of a laboratory manual are admirable, and it is only to be regretted that the subject matter is not well enough balanced to make the book generally applicable.

J. H. MORECROFT.

Electric Central Station Distributing Systems. By H. B. GEAR and P. F. WILLIAMS, New York, D. Van Nostrand Co., 1911. $3.00.

Messrs. Gear and Williams have written a book on the subject of distributing systems which will be found very useful, not only to enable the general practitioner to brush up for special occasions, but also for engineers of other professions and students in technical schools who wish to become familiar with the methods and principles of electrical distribution.

The book is written in a clear, simple style, requiring very little understanding of mathematics or higher physics, and gives just enough of the principles to make clear the reason for the methods and practices described. It is rather practical in scope and descriptive in character, and will be found of value particularly to men of various professions who have charge of engineering properties which include an electrical power station and distributing system.

Much practical data and information are given on the subjects of poles, lines, insulators and accessories, and descriptions of commercial types of transformers, voltage regulators and protective apparatus are included.

W. I. SLICHTER.

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VOL. XXXIII. NO. 2. JANUARY, 1912

THE

SCHOOL OF MINES QUARTERLY.

A JOURNAL OF APPLIED SCIENCE.

CONTENTS

	PAGE
THEORY OF ELASTIC ARCH RIBS. By *William H. Burr*.....	109
STUDY OF MECHANICAL CONVEYORS—I. By *Lincoln DeG. Moss*	136
COURSES IN HIGHWAY ENGINEERING AT COLUMBIA. By *Arthur H. Blanchard*	155
GEOLOGY OF THE ZARUMA MINES, ECUADOR. By *F. G. Baragwanath*	160
HAN-YEH-PING IRON AND COAL COMPANY, CHINA. By *Cho Yang and Cheng-Fu Wang*.....	165
ABSTRACTS—ANALYTICAL CHEMISTRY. By *E. Waller*.....	173
DEPARTMENTAL REPORTS FOR 1910-11.....	194
BOOK REVIEWS	204
PERSONAL ITEMS	207

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THE GENERAL THEORY OF ELASTIC ARCH RIBS ACCORDING TO THE LAW OF LEAST WORK.

BY WM. H. BURR.*

ARTICLE I. GENERAL CONSIDERATIONS ON THE THEORY OF ARCHED RIBS.

An arch rib differs from an ordinary bridge truss chiefly in that the reactions at the two ends of the rib are not vertical, a horizontal component always existing. The arch rib, like the truss, may be continuous, but that continuity may be such as to join the ends of the rib to the piers only which support them. The ends of the arch rib may also be hinged or jointed and again there may be a hinge or joint at the crown. If the hinge at the crown is absent the arch rib is statically indeterminate whether its ends are hinged or continuous, but if the rib has hinges both at the ends and at the crown it is statically determinate, requiring only the ordinary statical equations of equilibrium for the complete analysis of its stresses.

In the cases of statical indetermination, i.e., when the ends are continuous or, when there are hinges at the ends only, it is necessary to employ equations of condition which depend upon the elastic properties of the materials used in the rib, in addition to the ordinary statical equations of equilibrium. These equations may be based upon others giving deflections by the common theory of flexure, as in the case of stiffened suspension bridges, or they may be

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found by the aid of the law or principal of least work, also as illustrated in the theory of suspension bridges. In fact, an arch rib is essentially an inverted, stiffened, suspension bridge, by which transformation the stiffened cable becomes the arch rib in compression, the bending moments in the stiffening truss or stiffening bracing of the suspension bridge corresponding to the bending moments of the arch rib. It is clear, therefore, that the arch rib must sustain a combination of bending or flexure and direct compression, and its cross-section must be designed accordingly. That cross-section may be solid, like either a plate girder or a plain concrete arch of a single material, or a solid reinforced concrete section; or again, the rib may be an articulated structure with open bracing, as in many steel arch ribs for railway station buildings and other similar structures. It is equally clear that the cross-section of the arch rib may be subjected to transverse shear in a normal cross-section or to a shear in some other direction, depending upon the character of the loading.

Arch ribs may be subjected to loading in any direction, i.e., normal to the exterior surface, as in the case of wind pressure, or in a vertical direction, as in the case of many plain and reinforced concrete arches, or in any other direction. A proper theory for such structures must therefore provide for loading in any direction and for the effects of bending moments, normal stresses and shears.

In the following articles will be given a general analysis for any conditions of loading, based upon both the principle of least work and upon elastic deflection.

Article 2. Arch Rib with Fixed Ends — Stresses and Moments — Total Work Expended.

The general treatment of an arch rib with fixed ends presupposes any curve for the rib and any system of loads, as illustrated in Fig. 1. The loads P_1, P_2, P_3, etc., acting at any points and in any directions, are shown as applied to the center line of the rib having the span F K or l. The points F and K are supposed to be at the same elevation and F is the origin of the coördinates x and y of the curved axis of the rib.

The points c and c' represent any normal sections of the rib, with which the lines of action of P_1, P_2, etc., make the respective

angles ϕ_1, ϕ_2, etc., the normal sections themselves making the angles ϕ with a horizontal plane. The lines of action of P_1, P_2, etc., make the angles α_1, α_2, etc., with the horizontal so that:

$$\alpha_1 + \phi_1 = \alpha_2 + \phi_2 = \text{etc.} = \phi.$$

It is to be carefully observed that α_1, α_2, etc., ϕ_1, ϕ_2, etc., are positive as measured clockwise, i.e., right handed from a horizontal line or from the lines of action of P_1, P_2, etc. The same observation applies to the angles ϕ locating the normal sections of the rib and measured clockwise from the same horizontal line as α_1, α_2, etc.

It will first be necessary to determine the total normal and tangential (shearing) forces or stresses in each normal section of the rib. The requisite expression may readily be written as follows, H and V being the horizontal and vertical components respectively of the reaction at F, while H′ and V′ will be the corresponding components at K.

The component of any load P_1, Fig. 1, acting perpendicular to any normal section c or c' of the rib, is $P_1 \sin \phi_1$, and the tangential or shearing component of the same load is $P_1 \cos \phi_1$. Similarly the horizontal and vertical components of the reaction at F will have normal and tangential components, respectively, H $\sin \phi$, H $\cos \phi$, and V $\cos \phi$, $-$V $\sin \phi$. If the angle made by any force P_1 with the normal section is negative, like $-\phi_3$ for P_3, the normal component becomes $-P_3 \sin \phi_3$, although the tangential component may still remain $+ P_3 \cos \phi_3$.

As the forces and moments to which any part Fc or Fc' of the arch is subjected are being considered, a positive value of the normal force acting on any joint c or c' indicates compression and a negative force, tension.

The moment about the joint c or c' of every load or force acting on any part Fc or Fc' of the rib must also be written. For this purpose let a_1 and b_1 be the horizontal and vertical coördinates of the point of application of any force or load on the axis of the arch, and let x and y be the corresponding coördinates of the normal section c or c'. Then the lever arm of any such force P_1 will be $(y-b_1) \cos \alpha_1 + (x-a_1) \sin \alpha_1$.

Hence the desired moment for P_1 will be:

$$P_1 \left\{ (y - b_1) \cos \alpha_1 + (x - a_1) \sin \alpha_1 \right\} \ldots\ldots\ldots\ldots\ldots\ldots 1$$

For H, this expression becomes, since $a_1 = 0$, and $a_1 = b_1 = 0$:

$$H\,y \qquad \ldots\ldots 2$$

And for V, since $a_1 = -90°$ and $a_1 = b_1 = 0$:

$$-V\,x \qquad \ldots\ldots 3$$

As the ends of the arch rib in this case are fixed, each of the end joints F and K is subjected to a bending moment of such magnitude and direction as is necessary to secure fixedness under any loading. In general the moments will not be equal nor of the same sign. Let this external or fixing moment acting at F be represented by M_0. The resultant bending moment at any section of the rib will then be the algebraic sum of M_0 and the ordinary non-continuous moment in the curved beam or rib acted upon by H, V. P_1, P_2, etc., as shown below in Eq. 6.

The following general expressions for the normal and tangential components of the external loads N and S on any normal section of the rib, and for the moments M of those loads or forces about the center of that normal section may now be written as follows:

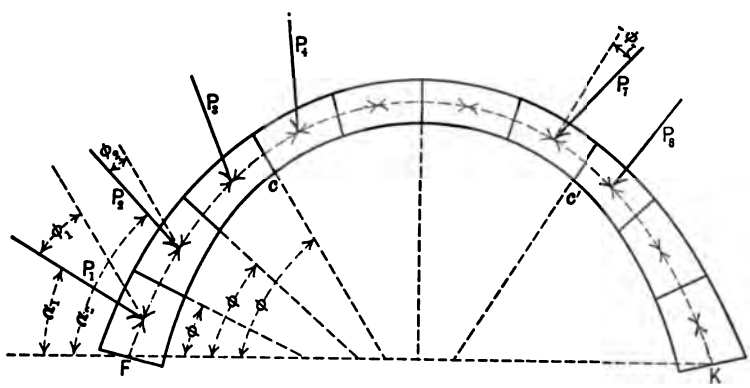


Fig. 1

$$N = H \sin\phi + V \cos\phi + \sum_0^{\phi} P_1 \sin\phi_1 \qquad \ldots\ldots 4$$

$$S = H \cos\phi - V \sin\phi + \sum_0^{\phi} P_1 \cos\phi_1 \qquad \ldots\ldots 5$$

$$M = M_0 + Hy - Vx + \sum_0^{\phi} P_1 \left\{ (y - b_1) \cos a_1 + (x - a_1) \sin a_1 \right\} \qquad 6$$

These equations are obviously perfectly general and they recognize the radial shear S.

In all that follows, there will be used:

$$M^1=\sum_0^{\phi} P_1\left\{(y-b_1)\cos a_1+(x-a_1)\sin a_1\right\} \quad \ldots\ldots\ldots 6a$$

In accordance with the law of least work, H, V and M_0 must have such values that the total work W expended in stressing all the members of the arch rib will be a minimum. The expression for that total expended work is, if L is the length of the rib measured along its axis:

$$W^*=\frac{1}{2}\int\frac{M^2}{EI}dL+\frac{1}{2}\int\frac{N^2}{EA}dL+\frac{6}{5}\int\frac{S^2}{GA'}dL=$$
$$\frac{1}{2E}\left(\int\frac{M^2}{I}dL+\int\frac{N^2}{A}dL\right)+\frac{6}{5G}\int\frac{S^2}{A'}dL \quad \ldots\ldots 7$$

In this equation, I is the moment of inertia of any normal section of the rib about its neutral axis, while A is the area of that cross-section and A′ is the normal sectional area on which the radial shear S acts. The area A may or may not be equal to A′. Each dL is an indefinitely short part of the rib or of the member carrying the shear S. G is the modulus of elasticity for shear and E, that for direct stress.

If the rib is of rectangular cross-section and of plain concrete, *b* being the breadth of the rib and *h* the depth, $A'=bh$ and S will be the total radial shear (Eq. 5) at any normal section. If the concrete be reinforced, A′ will have the same value *bh* plus such allowance as may be made for the steel part of the section. The latter may be taken in some cases as an area of concrete section sufficient in amount to afford shearing resistance equivalent to that of the steel.

If the rib should be of steel plate girder section, A′ would probably be taken as the area of the normal section of the web plate, possibly allowing for the rivet holes.

Whenever the shear section A′ is constant, we may at once write:

$$\frac{6}{5G}\int\frac{S^2}{A}dL=\frac{6}{5GA'}\int S^2\,dL \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots 8$$

* "Elasticity and Resistance of Materials," by Wm. H. Burr, Art. 29.

If the arch rib is an articulated truss with the usual chords and web members, as in Fig. 2, the expression for the work of that shear S in Eqs. 7 and 8 must again be modified. If there be either

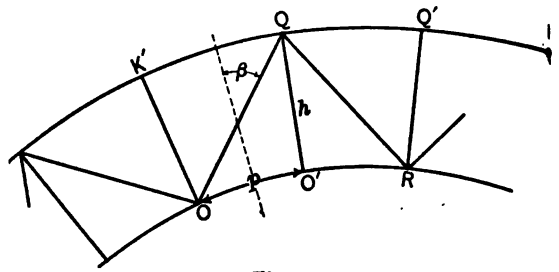


Fig. 2

a single or a double system of web members, the shear S remains constant throughout the panel length $p = O\ O'$. The tension or compression in the web member O Q will then he S *sec* β. The length of the web member O Q will be p *cosec* β. Hence if A″ is the area of the normal cross-section of the web member O Q, the work expended in stressing it will be:

$$W'' = \frac{S^2 \sec^2 \beta \operatorname{cosec} \beta}{2EA''} p \quad \ldots\ldots 9$$

If there are two systems of bracing, Eq. 9 will be in no way changed, but A″ will then represent the area of the normal cross-section of the two web members found in the panel p.

If the system of web bracing is diagonal and radial, as O Q O′ Q′, Fig. 2, p. in Eq. 9 should be displaced by $h \tan \beta$, after which β is to be made zero. Introducing these changes in Eq. 9, the work expended in stressing the radial web member O′ Q will be:

$$W''' = \frac{S^2 h}{2EA'''} \quad \ldots\ldots 10$$

Obviously, A‴′ is the area of normal section of the radial web member.

In the general application of Eq. 7, the third term of the second member of that equation is to be displaced by Eq. 9 for each part of the integration covering a panel p, S being constant for each panel but varying from panel to panel.

If diagonal and radial bracing is used in the rib the sum of the second members of Eqs. 9 and 10 must displace the third term of the second member of Eq. 7, as explained above, instead of that of Eq. 9 alone.

ARTICLE 3. ARCH RIBS WITH FIXED ENDS — APPLICATION OF PRINCIPLE OF LEAST WORK.

It is stated in connection with Eq. 7 of the preceding Article that H, V and M_0 must have values which will satisfy the law of least work. This requires that the first partial derivatives of W (Eq. 7), in respect to H, V and M_0 as independent variables, shall each be zero. Hence, displacing the integral sign by the more general summation sign, and making $G = nE$, Eq. 7 will give:

$$\frac{\delta W}{\delta H} = \frac{dL}{E}\left\{\sum \frac{M}{I}\frac{\delta M}{\delta H} + \sum \frac{N}{A}\frac{\delta N}{\delta H} + \frac{12}{5n}\sum \frac{S}{A'}\frac{\delta S}{\delta H}\right\} \quad \ldots 11$$

$$\frac{\delta W}{\delta M_0} = \frac{dL}{E}\sum \frac{M}{I}\frac{\delta M}{\delta M_0} \quad \ldots 12$$

$$\frac{\delta W}{\delta V} = \frac{dL}{E}\left\{\sum \frac{M}{I}\frac{\delta M}{\delta V} + \sum \frac{N}{A}\frac{\delta N}{\delta V} + \frac{12}{5n}\sum \frac{S}{A'}\frac{\delta S}{\delta V}\right\} \quad \ldots 13$$

The quantity n (the ratio $\frac{G}{E}$) is known.

In the actual use of these equations the quantity dL will have finite values each representing the length of a section of the rib, or some other finite length.

Again, the third term in the second members of Eqs. 11 and 13 will be modified for particular cases as already set forth in connection with Eqs. 9 and 10, but without changing the principal elements of those terms.

By differentiating Eqs. 4, 5, and 6:

$$\frac{\delta M}{\delta H} = y. \qquad \frac{\delta M}{\delta V} = -x.$$

$$\frac{\delta N}{\delta H} = \sin\phi \qquad \frac{\delta N}{\delta V} = \cos\phi$$

$$\frac{\delta S}{\delta H} = \cos\phi \qquad \frac{\delta S}{\delta V} = -\sin\phi$$

$$\frac{\delta M}{\delta M_0} = 1$$

These values are to be inserted in Eqs. 11, 12, and 13. Then writing ΔL for dL and extending the summations over the entire length L of the rib, each of those equations must be placed equal to zero, remembering that E divides out and disappears:

$$\left[\sum\frac{M}{I}y+\sum\frac{N}{A}\sin\phi+\frac{12}{5n}\sum\frac{S}{A'}\cos\phi\right]_0^L\Delta L=0 \quad \ldots\ldots 14$$

$$\sum_0^L\frac{M}{I}\Delta L=0 \quad \ldots\ldots 15$$

$$\left[-\sum\frac{M}{I}x+\sum\frac{N}{A}\cos\phi-\frac{12}{5n}\sum\frac{S}{A'}\sin\phi\right]_0^L\Delta L=0 \quad \ldots\ldots 16$$

The values of N, S, and M given in Eqs. 4, 5 and 6 must next be substituted in Eqs. 14, 15 and 16 giving three equations from which H, V and M_0 may at once be found.

To simplify the forms of those equations let the following notation be used.

$$a=-\frac{\left[\sum\frac{yx}{I}-\sum\frac{\sin\phi\cos\phi}{A}+\frac{12}{5n}\sum\frac{\sin\phi\cos\phi}{A'}\right]\Delta L}{\left(\sum\frac{y^2}{I}+\sum\frac{\sin^2\phi}{A}+\frac{12}{5n}\sum\frac{\cos^2\phi}{A'}\right)\Delta L=B} \quad \ldots\ldots 17$$

$$b=\frac{\sum\frac{y}{I}\Delta L}{B} \quad \ldots\ldots 18$$

$$c=\frac{\left[\sum\frac{M'y}{I}+\sum\frac{\sin\phi}{A}\left(\sum_0^\phi P_1\sin\phi_1\right)+\frac{12}{5n}\sum\frac{\cos\phi}{A'}\left(\sum_0^\phi P_1\cos\phi_1\right)\right]\Delta L}{B} \quad \ldots\ldots 19$$

$$a'=-\frac{\sum\frac{x}{I}\Delta L}{\sum\frac{y}{I}\Delta L};\quad b'=\frac{\sum\frac{\Delta L}{I}}{\sum\frac{y}{I}\Delta L};\quad c'=-\frac{\sum\frac{M'}{I}\Delta L}{\sum\frac{y}{I}\Delta L} \quad \ldots\ldots 20$$

$$a''=-\frac{\left[\sum\frac{x^2}{I}+\sum\frac{\cos^2\phi}{A}+\frac{12}{5n}\sum\frac{\sin^2\phi}{A'}\right]\Delta L}{\left(\sum\frac{xy}{I}-\sum\frac{\sin\phi\cos\phi}{A}+\frac{12}{5n}\sum\frac{\sin\phi\cos\phi}{A'}\right)\Delta L=C} \quad 21$$

$$b''=\frac{\sum\frac{x}{I}\Delta L}{C} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots 22$$

$$c''=\frac{\left[\sum\frac{M'x}{I}-\sum\frac{\cos\phi}{A}\left(\sum_0^{\phi}P_1\sin\phi_1\right)+\frac{12}{5n}\sum\frac{\sin\phi}{A'}\left(\sum_0^{\phi}P_1\cos\phi_1\right)\right]\Delta L}{C} \quad \ldots\ldots 23$$

The values of B and C are obvious as shown, and M′ is given by Eq. 6a.

When the parts into which the rib may be divided, as indicated in Fig. 1, are all equal, ΔL will divide out and disappear from Eqs. 17, 18, 19, 20, 21, 22 and 23; but when, as in the general case, those parts are unequal, ΔL must be made a factor of each quantity preceded by the summation sign Σ, as shown in Eqs. 18, 20 and 22, for ΔL is then a variable quantity and of finite value in the actual application of the formulae.

Making the substitutions indicated above, and using the notation shown, the following equations will result:

$$H+a\,V+b\,M_0+c=0 \quad \ldots\ldots\ldots\ldots\ldots\ldots 24$$
$$H+a'\,V+b'\,M_0+c'=0 \quad \ldots\ldots\ldots\ldots\ldots\ldots 25$$
$$H+a''\,V+b''\,M_0+c''=0 \quad \ldots\ldots\ldots\ldots\ldots\ldots 26$$

Subtracting Eq. 24 from Eq. 25, and then subtracting Eq. 25 from Eq. 26:

$$V(a'-a)+M_0(b'-b)+c'-c=0 \quad \ldots\ldots\ldots 27$$
$$V(a''-a')+M_0(b''-b')+c''-c'=0 \quad \ldots\ldots 28$$

Eliminating V in a similar manner by using Eqs. 27 and 28:

$$M_0=\frac{(c'-c)\frac{a''-a'}{a'-a}-(c''-c')}{(b''-b')-(b'-b)\frac{a''-a'}{a'-a}} \quad \ldots\ldots\ldots\ldots\ldots\ldots 29$$

Eq. 27 will then give:

$$V=M_0\frac{b-b'}{a'-a}-\frac{c'-c}{a'-a} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots 30$$

Or, by Eq. 28:

$$V=M_0\frac{b'-b''}{a''-a'}-\frac{c''-c'}{a''-a'} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots 31$$

Then by Eq. 24 (Eq. 25 or Eq. 26 may be similarly used):

$$H = -aV - bM_0 - c \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots 32$$

In Eqs. 14 to 23, both inclusive, the summation sign Σ is to be taken as covering the entire rib, i.e., $\sum_0^L$ except in terms where the limits ϕ and o are shown, as in two terms each of Eqs. 19 and 23.

In this latter case, however, the summation

$$\sum \frac{cos\,\phi}{A}\left(\sum_0^{\phi} P_1\, sin\phi_1\right)$$

is to be taken between the limits a_1 (the point of application of the force or load nearest to the left end of the span) and l, the right end of the span. In fact, the summation for each load is to extend from its point of application to the right end of the span. This procedure will be illustrated later by actual application.

The preceding equations are perfectly general for any loading whatever and for any curve of rib, whether symmetrical or unsymmetrical. They can be materially simplified for symmetrical curves and vertical loading.

ART. 4. ARCH RIBS WITH FIXED ENDS — SYMMETRICAL ARCHES

When the arch curve is symmetrical in reference to a vertical line through the crown, some of the equations 17 to 23 of the preceding Article may be materially simplified.

In this case the joints located by the angle ϕ^1, Fig. 1, or in general by ϕ, will be symmetrically placed in reference to the crown of the arch, whether their number be even or odd. Then, since $cos\ (90° - \phi) = -cos\ (90° + \phi)$ and $sin\ (90° - \phi) = sin\ (90° + \phi)$, and remembering that for symmetrical ribs A, A′ and ΔL have each the same value at equal distances either side of the crown:

$$\sum_o^l \frac{sin\phi\ cos\phi}{A}\Delta L = 0 = \sum_o^l \frac{sin\phi\ cos\phi}{A'}\Delta L.$$

The summation limits l and o have the same significance as L and o.

This result reduces the numerator of the second member of Eq. 17, and the denominator of the second member of Eq. 21, to the single term, $\sum_o^l \frac{yx}{I}\Delta L$.

Furtherfore, this latter summation is composed of pairs of terms of the kind;

$$\left(\frac{yx}{I}+\frac{(l-x)y}{I}\right)\Delta L = l\frac{y}{I}\Delta L$$

Hence:

$$\sum_0^l \frac{yx}{I}\Delta L = l\sum_0^{\frac{l}{2}} \frac{y\Delta L}{I} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots 33$$

It is also evident, for these conditions of symmetry, that:

$$\sum_0^l \frac{y^2}{I}\Delta L = 2\sum_0^{\frac{l}{2}} \frac{y^2}{I}\Delta L; \sum_0^l \frac{\sin^2\phi}{A}\Delta L = 2\sum_0^{\frac{l}{2}} \frac{\sin^2\phi}{A}\Delta L;$$

and

$$\sum_0^l \frac{\cos^2\phi}{A'}\Delta L = 2\sum_0^{\frac{l}{2}} \frac{\cos^2\phi}{A'}\Delta L \quad \ldots\ldots\ldots\ldots\ldots\ldots 34$$

In Eqs. 18 and 20, obviously:

$$\sum_0^l \frac{y}{I}\Delta L = 2\sum_0^{\frac{l}{2}} \frac{y\Delta L}{I} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots 35$$

The summation in the numerator of the first of Eqs. 20 is composed of pairs of terms:

$$\frac{x\Delta L}{I}+\frac{(l-x)\Delta L}{I} = l\frac{\Delta L}{I}$$

Hence:

$$\sum_0^l \frac{x\Delta L}{I} = l\sum_0^{\frac{l}{2}} \frac{\Delta L}{I} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots 36$$

Finally:

$$\sum_0^l \frac{\Delta L}{I} = 2\sum_0^{\frac{l}{2}} \frac{\Delta L}{I} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots 37$$

Introducing these modifications in Eqs. 17 to 23, including both, these will result:

For Symmetrical Arches with Symmetrical Ribs:

$$a=-\frac{l}{2}\frac{\sum_0^{\frac{l}{2}}\frac{y\Delta L}{I}}{\left(\sum_0^{\frac{l}{2}}\frac{y^2\Delta L}{I}+\sum_0^{\frac{l}{2}}\frac{\sin^2\phi\Delta L}{A}+\frac{12}{5n}\sum_0^{\frac{l}{2}}\frac{\cos^2\phi\Delta L}{A'}\right)=B_1} \quad \ldots\ldots 38$$

$$b=\frac{\sum_0^{\frac{l}{2}}\frac{y\Delta L}{I}}{B_1} \quad \ldots\ldots 39$$

$$c=\frac{\sum_0^{l}\frac{M'y\Delta L}{I}+\sum_0^{l}\frac{\sin\phi}{A}\left(\sum_0^{\phi}P_1\sin\phi_1\right)\Delta L+\frac{12}{5n}\sum_0^{l}\frac{\cos\phi}{A'}\left(\sum_0^{\phi}P_1\cos\phi_1\right)\Delta L}{2B} \quad \ldots\ldots 40$$

$$a'=-\frac{l}{2}\frac{\sum_0^{\frac{l}{2}}\frac{\Delta L}{I}}{\sum_0^{\frac{l}{2}}\frac{y\Delta L}{I}};\quad b'=\frac{\sum_0^{\frac{l}{2}}\frac{\Delta L}{I}}{\sum_0^{\frac{l}{2}}\frac{y\Delta L}{I}};\quad c'=\frac{\sum_0^{l}\frac{M'\Delta L}{I}}{2\sum_0^{\frac{l}{2}}\frac{y\Delta L}{I}} \quad \ldots 41$$

$$a''=\frac{\sum_0^{l}\frac{x^2\Delta L}{I}+2\sum_0^{\frac{l}{2}}\frac{\cos^2\phi\Delta L}{A}+\frac{24}{5n}\sum_0^{\frac{l}{2}}\frac{\sin^2\phi\Delta L}{A'}}{l\sum_0^{\frac{l}{2}}\frac{y\Delta L}{I}} \quad \ldots 42$$

$$b''=\frac{\sum_0^{\frac{l}{2}}\frac{\Delta L}{I}}{\sum_0^{\frac{l}{2}}\frac{y\Delta L}{I}}=b' \quad \ldots\ldots 43$$

$$c''=\frac{\sum_0^{l}\frac{M'x\Delta L}{I}-\sum_0^{l}\frac{\cos\phi}{A}\left(\sum_0^{\phi}P_1\sin\phi_1\right)\Delta L+\frac{12}{5n}\sum_0^{l}\frac{\sin\phi}{A'}\left(\sum_0^{\phi}P_1\cos\phi_1\right)\Delta L}{l\sum_0^{\frac{l}{2}}\frac{y\Delta L}{I}} \quad \ldots\ldots 44$$

These equations are applicable for any loading acting in any direction; it is only required that the points of loading shall be symmetrically placed.

As $b''=b'$, Eq. 29 gives:

$$M_0=\frac{\frac{a'-a}{a''-a'}\left(c''-c'\right)-c'+c}{b'-b} \quad \ldots\ldots 44a$$

Also, by Eq. 31:

$$V=\frac{c'-c''}{a''-a'} \quad \ldots\ldots 44b$$

The value of H given by Eq. 32 remains unchanged.

It is to be carefully observed in applying the preceding equations that $\sum_0^{\phi}P_1\sin\phi_1$ is the total, normal, external force or loading on any joint located by the angle ϕ; also, that $\sum_0^{\phi}P_1\cos\phi_1$ is the total external shearing force or loading on the same joint. A similar observation applies to M′ (Eq. 6a). By referring to Eqs. 4, 5 and 6, in connection with the procedures by which Eqs. 14, 15

and 16 were established, it will be clear that the summations involving M′ or P_1 such as $\sum_0^l \frac{\cos\phi}{A}\left(\sum_0^\phi P_1 \sin\phi_1\right)\Delta L$ are to be extended from the point of application only of each P_1 to l. There must be in effect, therefore, if not in general form, a summation for each load P_1, P_2, etc., beginning at its point of application and extending to the further end of the span.

Vertical Loading.

Most reinforced arches are not only symmetrical, but their loading is usually vertical, making $a_1 = a_2 = \text{etc.} = 90°$ and $90° + \phi_1 = \phi$ or $\phi_1 = \phi - 90°$. Hence for that condition:

$$\sin\phi_1 = -\cos\phi \text{ and } \cos\phi_1 = \sin\phi \quad \ldots\ldots\ldots 44\,c$$

These values may conveniently be used in Eqs. 40 and 44, but no other changes are required for vertical loading.

In consequence of Eq. 44c, if in Eqs. 40 and 44 there is but a single force P_1.

$$\sum_0^l \frac{\sin\phi}{A}\left(\sum_0^\phi P_1 \sin\phi_1\right)\Delta L = -P_1\sum_{a_1}^l \frac{\sin\phi\cos\phi}{A}\Delta L \cdot 44\,d$$

$$\sum_0^l \frac{\cos\phi}{A'}\left(\sum_0^\phi P_1 \cos\phi_1\right)\Delta L = P_1\sum_{a_1}^l \frac{\sin\phi\cos\phi}{A'}\Delta L \cdot \ldots 44\,e$$

$$\sum_0^l \frac{\cos\phi}{A}\left(\sum_0^\phi P_1 \sin\phi_1\right)\Delta L = -P_1\sum_{a_1}^l \frac{\cos^2\phi}{A}\Delta L \ldots 44\,f$$

$$\sum_0^l \frac{\sin\phi}{A'}\left(\sum_0^\phi P_1 \cos\phi_1\right)\Delta L = P_1\sum_{a_1}^l \frac{\sin^2\phi}{A'}\Delta L \ldots\ldots 44\,g$$

Furthermore, since $\cos(90° - \phi) = -\cos(90° + \phi)$, the summation $\sum \frac{\sin\phi\cos\phi}{A}\Delta L$ between the symmetrical limits a_1 and $l - a_1$ will be zero. Again, because $\cos(180 - \phi) = -\cos\phi$ there will result for any load for which a_1, a_2, etc., is less than $\frac{1}{2}l$:

$$-P_1\sum_{a_1}^{l}\frac{\sin\phi\cos\phi}{A}\Delta L=P_1\sum_{0}^{a_1-1}\frac{\sin\phi\cos\phi}{A}\Delta L \quad ..44h$$

For the same reason:

$$P_1\sum_{a_1}^{l}\frac{\sin\phi\cos\phi}{A'}\Delta L=-P_1\sum_{0}^{a_1-1}\frac{\sin\phi\cos\phi}{A'}\Delta L \quad ..44k$$

Again, when a_1, a_2, etc., is greater than $\frac{1}{2}\,l$, the summation of Eq. 44d is also expressed by Eq. 44h, since the symmetrical terms of the summation (symmetrical in reference to the crown of the arch) cancel each other in pairs, as already observed.

For the same reason, when a_1, a_2, etc., is greater than $\frac{1}{2}\,l$, the summation of Eq. 44e is expressed by Eq. 44k.

In the use of Eqs. 44*h* and 44*k*, it must be carefully remembered that ϕ is an acute angle. With the vertical loads supposed, Eq. 6*a* shows: $M'=\sum_{0}^{\phi'}P_1(x-a_1)$ 44*l*

If a single load P_1 rests on the arch the term $\sum_{0}^{l}\frac{M'y\,\Delta L}{I}$ in Eq. 40 may be written:

$$\sum_{0}^{l}\frac{M'y\,\Delta L}{I}=P_1\sum_{a_1}^{l}\frac{(x-a_1)\,y\Delta L}{I}\quad .....44\,m$$

The second member of Eq. 44m is to be used in computations to be made hereafter.

Under the same supposition of a single vertical load the numerator of Eq. 41 becomes:

$$\sum_{0}^{l}\frac{M'\Delta L}{I}=P_1\left(\sum_{a_1}^{l}\frac{x\,\Delta L}{I}-a_1\sum_{a_1}^{l}\frac{\Delta L}{I}\right)\quad .....45$$

These equations will greatly simplify and shorten the treatment of symmetrical arch ribs with vertical loading, as will be shown further on.

ART. 5. MOMENT OF INERTIA — INTENSITY OF STRESS.

Moment of Inertia, I

If the rib is of two materials, like reinforced concrete, the moment of inertia I must be written accordingly.

In order to write the value of I for a reinforced concrete rib, let A_2 represent the area of the steel section divided into the two parts $r\,A_2$ in the lower half of the rib and $(1-r)\,A_2$ in the upper half. Assuming the cross-section of the rib to be rectangular, let the dimensions of that cross-section be as shown in Fig. 3, the line

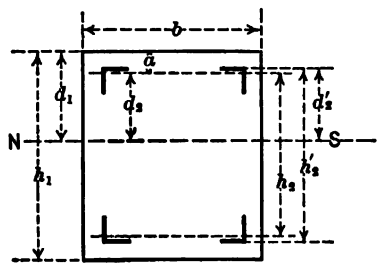


Fig. 3

N S being the neutral axis, or line of no stress, of the section. The notation of Fig. 3 is self explanatory, but it may be well to state that d_2 is the distance from the neutral axis to the center of gravity of $(1-r)\,A_2$, while $h_2 - d_2$ is the distance from the neutral axis to the center of gravity of $r\,A_2$. If, then, the coefficient or modulus of elasticity of steel be represented by E_2 while E represents that of concrete, and if we write

$$\frac{E_2}{E} = e \qquad \ldots\ldots 46$$

the distance d_1 from the upper surface of the rib to the neutral axis will take the value:

$$d_1 = \frac{\frac{1}{2}h_1^2 + e\frac{A_2}{b}(a + rh_2)}{h_1 + e\frac{A_2}{b}} \qquad \ldots\ldots 47\,*$$

In this equation the tensile resistance of the concrete is given full value.

If the tensile resistance of the concrete be neglected, the value of d_1 becomes:

* "Elasticity and Resistance of Materials," by Wm. H. Burr, pp. 639, 640.

$$d_1 = -e\frac{A_2}{b} \pm \sqrt{\left(e\frac{A_2}{b}\right)^2 + 2e\frac{A_2}{b}\left(a + rh_2\right)} \quad \ldots\ldots\ldots\ldots\ldots 48 *$$

If the steel should be found wholly on one side or the other (i.e., top or bottom) of the rib, $r = 0$ or 1 as the case may be.

Similar formulae may be written for other forms of cross-section than the rectangle.

It is obvious that if the cross-section of both steel and concrete is symmetrically disposed about a horizontal line at mid-depth of the rib, the neutral axis will coincide with that mid-depth line, making,

$$d_1 = \tfrac{1}{2} h_1$$

This is shown by Eq. 47 because r is then $\frac{1}{2}$ and $a + \frac{1}{2} h_2 = \frac{1}{2} h_1$.

After d_1 has been determined, the moment of inertia I of the combined cross-section about the neutral axis, now known, must be found.

If the radius of curvature due to bending only is R; then it must be remembered that the moment of the internal stresses of any normal section is:

$$M = \frac{EI}{R},$$

for each part of the section. Hence, the quantity E I for the reinforced concrete section may be written:

$$E I + E_2 I_2 = E (I + e I_2) \quad \ldots\ldots\ldots\ldots\ldots 49$$

It is convenient to compute I as if it covered the entire normal section, i.e., not only the concrete part but the area of the steel section; thus giving to Eq. 49 the following form:

$$E I + E_2 I_2 = E [I_1 + (e - 1) I_2] \quad \ldots\ldots\ldots\ldots 50$$

Eqs. 49 and 50 are perfectly general, and they are applicable to ribs of any section.

In Eq. 50, I_1 is the moment of inertia of the entire cross-section of the rib about the neutral axis.

If the width or breadth of a rectangular rib is b, and h_1 is its depth, then if $d_1 = \frac{1}{2} h_1$:

$$I_1 = \frac{b h_1^3}{12} \quad \ldots\ldots\ldots\ldots\ldots 51$$

Assuming, as is usually the case, that the steel is symmetrically disposed, these may be written:

* *Loc. cit.*

$$I_2 = 2\left(\frac{A_2}{2} d\right) = A_2 d_2^2 \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots 52$$

Inasmuch as the area of steel seldom exceeds 2 percent. of the entire section, and is often as small as ½ percent., each part of it may be considered concentrated at its center of gravity. This greatly simplifies finding the value of I.

The moment of inertia of the combined rectangular section then becomes:

$$I = I_1 + \left(e - 1\right) I_2 = \frac{b h_1^3}{12} + \left(e - 1\right) A_2 d_2^2 \quad \ldots\ldots\ldots\ldots\ldots\ldots 53$$

If these equations are applied to one unit (foot) width of rib, $b = 1$ and A_2 is the area of normal section of steel belonging to a unit width of rib.

These formulae enable all computations of stresses in any reinforced concrete rib to be made when the bending moment M is known.

In the case of a plain concrete rib, the moment of inertia I is simply as given in Eq. 51. If the rib is of structural steel, the moment of inertia I is to be found precisely as for any steel cross-section, for which the neutral axis passes through its center of gravity.

Intensities of Stress.

Expressions for the intensities of stress in either the concrete or the steel may readily be written. For this purpose,

Let k_1 = (greatest) intensity of stress in the concrete at distance d_1 from the neutral axis.

k^1 = (greatest) intensity of stress in the concrete at distance $(h_1 - d_1)$ from the neutral axis.

c = (greatest) intensity of stress in the steel at distance d_2 from the neutral axis.

The sign of the bending moment M, Eq. 6, Art. 2, will show whether k^1, k_1 or c is tension or compression. A negative moment will make k_1 and c compression and, hence, k^1 tension.

The intensity k_1 will be:

$$k_1 = \frac{M}{\frac{b d_1^2}{3} + \frac{b (h_1 - d_1)^3}{3 d_1} + e \frac{d_2^2 A_2}{d_1}\left[(1 - r) + r\left(\frac{h_2}{d_2} - 1\right)^2\right]} \quad \ldots 54 *$$

then,

* *Op. cit.*, pp. 642, 643.

$$k' = \frac{h_1 - d_1}{d_1} k_1; \text{ and } c = e\frac{d_2}{d_1} k_1 \quad \ldots\ldots 55$$

Furthermore, the intensity of stress in the steel of the kind opposite to c is:

$$t_1 = \frac{h_2 - d_2}{d_2} c = e\frac{h_2 - d_2}{d_1} k_1 \quad \ldots\ldots 56$$

The preceding equations recognize the tensile resistance of the concrete, but if that tensile resistance be neglected, the intensity k_1, now compression, becomes:

$$k_1 = \frac{M}{\frac{bd_1^2}{3} + e\frac{d_2^2}{d_1} A_2 \left[(1-r) + r\left(\frac{h_2}{d_2} - 1\right)^2 \right]} \quad \ldots\ldots 57$$

In this case, $k^1 = 0$ but otherwise Eqs. 55 and 56 are unchanged.

Although the preceding equations belong to rectangular sections, it is clear that similar equations for any form of section may easily be written.

If the steel reinforcement is wholly on the tension side, $r = 1$ in Eq. 57, and:

$$k_1 = \frac{M}{\frac{bd_1^2}{3} + e\frac{A_2}{d_1}(h_2 - d_2)^2} \quad \ldots\ldots 58$$

Bending Moment, M.

In all the preceding equations, the moment M is given by Eq. 6, Art. 2, in which M_0, H and V are found by Eqs. 29, 30 and 32 of Art. 3.

When the loading is vertical, $a_1 = a_2 = \text{etc.} = 90°$, then $\cos a_1 = \cos a_2 = \text{etc.} = 0$ and $\sin a_1 = \sin a_2 = 1$. Hence:

$$M = M_0 + Hy - Vx + \sum_0^{\phi} P_1(x - a_1) \quad \ldots\ldots 58a$$

Normal Compressive Stress at Any Section.

The total normal stress N on any normal section of an arch rib, given by Eq. 4 of Art. 2, is uniformly distributed over the sectional area of the rib, except that in case of reinforced concrete the uniform intensity of the stress on the steel section A_2 is $e\left(=\frac{E_2}{E}\right)$ times as great as on the concrete section. Hence if p_1 is the uni-

form intensity of compression due to N, the direct normal stress, then A being the total normal sectional area of the rib:

$$p_1 A + (e - 1) p_1 A_2 = N.$$

Or:

$$p_1 = \frac{N}{A+(e-1)A_2} \quad \ldots\ldots 59$$

The greatest intensity of compression in the concrete section will then be, by Eqs. 54, 57 or 58 and 59:

$$p = k_1 + p_1 \quad \ldots\ldots 60$$

While the least intensity will be:

$$p^1 = k_1 - p_1 \quad \ldots\ldots 61$$

The greatest compression in the steel will be, by the aid of Eq. 55:

$$c = ep_1 + e\frac{d_2}{d_1}k_1 = \frac{eN}{A+(e-1)A_2} + e\frac{d_2}{d_1}k_1 \quad \ldots\ldots 62$$

The greatest tension in the steel in the same section will be, remembering Eq. 56:

$$t = ep_1 - e\frac{h_2-d_2}{d_1}k_1 = \frac{eN}{A+(e-1)A_2} - e\frac{h_2-d_2}{d_1}k_1 \quad \ldots\ldots 63$$

These equations complete the analysis for the direct stresses of compression and tension in any section of a reinforced concrete arch.

For plain concrete or any other single material it is only necessary to make $e = 1$ or $e - 1 = 0$ in the Eqs. 59, 62, and 63.

Vertical Loading.

If the rib carries vertical loading, $\alpha_1 = \alpha_2 = \text{etc.} = 90°$ and $\sin 90° = 1$, $\cos 90° = 0$. Also, as stated in the preceding Article, $\sin \phi_1 = -\cos \phi$ and $\cos \phi_1 = \sin \phi$. Hence, by Eq. 4 of Article 2:

$$N = H \sin\phi + \left(V - \sum_0^{\phi} P_1\right) \cos \phi \quad \ldots\ldots 64$$

This value of N is to be used in the preceding equations for p_1, c and t when the loading is vertical.

The bending moment M for vertical loading is given by Eq. 58 a.

Shearing Stress in Any Normal Section.

The general value of the total shearing stress S in any normal section of the arch rib is given by Eq. 5, Article 2. This shearing stress is assumed to be uniformly distributed over the normal sec-

tion A which includes the steel section A_2 in the case of reinforced concrete.

When the rib is of solid reinforced concrete it may seriously be doubted whether the steel should be considered as relieving the concrete of any material amount of shear unless the imbedded steel is designed as a braced rib in itself. If the steel is not so designed the intensity s of the shear in the concrete will be:

$$s=\frac{H\ cos\ \phi - V\ sin\ \phi + \sum_0^{\phi} P_1\ cos\ \phi_1}{A} \quad \ldots\ldots\ldots\ldots 65$$

The total sectional area A, which includes A_2, is written, as it is reasonable to suppose that the steel section is at least equivalent to an equal concrete section in resisting shear.

Should the steel reinforcement be designed as a trussed member or braced rib, the amount of shear which it will carry will depend upon that design to such an extent that the amount cannot be determined until the design is known; it may be designed to carry the total shear. The shear which the steel does not carry must evidently be carried by the concrete. It may be observed that it is difficult to arrange the combined concrete and steel so as to make each part carry satisfactorily its computed proper portion. In most cases it is probably best to design the work so that etither the concrete or the steel shall afford sufficient capacity to carry all the shear.

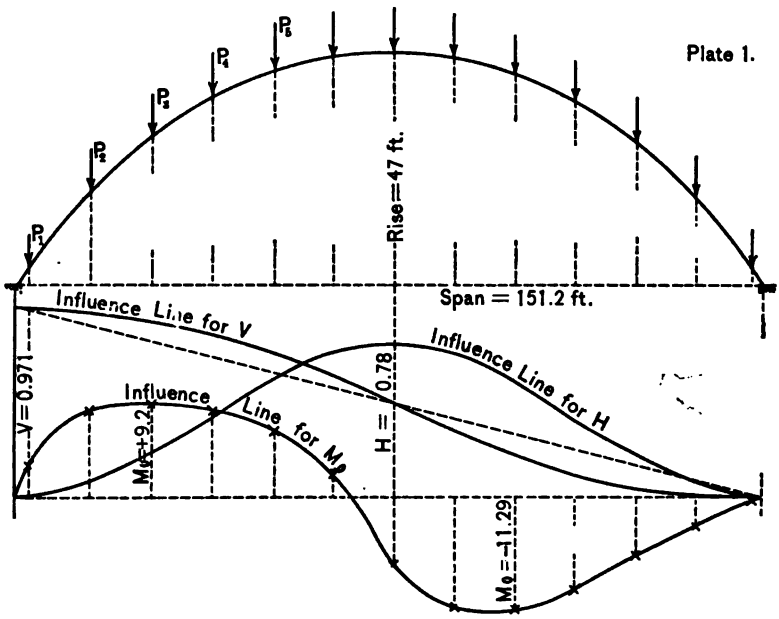
When all the loading is vertical, $cos\ \phi_1 = sin\ \phi$ in Eq. 65, or:

$$s=\frac{H\ cos\ \phi - sin\ \phi\ (V - \sum_0^{\phi} P_1)}{A} \quad \ldots\ldots\ldots\ldots\ldots 66$$

In some cases it may be advisable to determine the shear on some other than the normal section of the rib, as on a vertical plane normal to the vertical axial plane. It is only necessary to determine the resultant stress along such a plane in the same general manner as that employed for the normal section; then find the area of the section in question and divide the former by the latter, remembering the general observations regarding the steel reinforcement made above.

ART. 6. APPLICATION OF FORMULAE TO A REINFORCED CONCRETE RIB.

The rib used in the application of these formulae is shown on Pl. 1. It is taken from a bridge already constructed; the span of the rib from center to center of springing joints is 151.2 ft., and the rise



of the center line of the rib at the crown is 47 ft. The curve of the center-line of the rib is circular, but two centered, the radius of the central part being 78.2 ft. and that of each end part 106.5 ft. The uniform width of the rib is 3.25 ft. The depth of rib at the crown is 3.5 ft. and 7.5 ft. at the springing joint.

There are 13 loads applied to the rib by means of spandril columns, placed 12.25 ft. from center to center. The dead loads of the structure, and the moving loads are as follows:

Dead load.	*Live load.*	*Dead load.*	*Live load.*
P_1 = 75,600 lb.	lb.	P_8 = 36,500 lb.	24,500 lb.
P_2 = 53,300	24,500	P_9 = 44,000	24,500
P_3 = 49,800	24,500	P_{10} = 47,400	24,500
P_4 = 47,400	24,500	P_{11} = 49,800	24,500
P_5 = 44,000	24,500	P_{12} = 53,300	24,500
P_6 = 36,500	24,500	P_{13} = 75,600	24,500
P_7 = 36,500	24,500		

The points of application of the loads fix the lengths (ΔL) of

the sections into which the rib is divided. The first load P_1 acts on the rib at the horizontal distance $a_1 = 2.1$ ft. from the joint, where the center-line of the rib cuts the springing joint, which point is taken as the origin of the coördinates x and y, and a and b. The lengths ΔL along the center-line of the rib are the distances between the points of application of the loads P_1, P_2, P_3, etc., and the distances of P_1 and P_{13} from the ends of the center-line.

The coördinates x and y locate the normal sections of the rib at the mid-points of the length ΔL, *i.e.*, midway between the points of application of each two adjacent loads. This part of the procedure is based upon the tacit assumption that the state of stress is uniform throughout each length ΔL, and it involves a corresponding approximation.

All dimensions, even for cross-sectional areas of steel and for moments of inertia are taken in feet. Hence areas of sections will be given in square feet, and the modulus of elasticity for steel will be $144 \times 30{,}000{,}000 = 4{,}320{,}000{,}000$. The modulus for concrete must be similarly taken.

The subscript $_0$ applies to the springing joint, but A'_0 or I'_0 indicates the area of normal rib section or moment of inertia midway of the ΔL_0 at the end of the rib, and similarly for other letters affected with the prime and zero. The subscripts $_1$, $_2$, $_3$, etc., indicate the joints, loads, etc., numbered consecutively from the end of the rib.

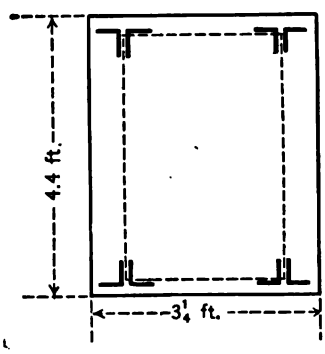


Fig. 4

Fig. 4 shows the cross-section of the rib with the steel reinforcement. The depth of the rib is represented by h_0, h'_0, h_1, h_2, etc., as given below. Fig. 4 shows the cross-section where the depth is $h_4 = 4.4$ feet.

The values of the angles ϕ_0, ϕ_0', ϕ', ϕ'', etc., of inclination of the normal section of the rib to a horizontal line at the mid-points of ΔL are:

$$\begin{aligned} \phi_0 &= 30^\circ\ 25' & \phi''' &= 56^\circ\ 45', \\ \phi_0' &= 32\ \ 00 & \phi^{iv} &= 66\ \ 45 \\ \phi' &= 38\ \ 35 & \phi^{v} &= 76\ \ 15 \\ \phi'' &= 48\ \ 15 & \phi^{vi} &= 85\ \ 05 \end{aligned}$$

The inclinations of the normal joints in the second half of the arch are not needed, as they would be supplements of the angles of inclination in the first half.

The lengths ΔL, the total areas of normal sections of the rib and the areas of steel section (about $\frac{1}{2}$ percent. of the total sectional area) are as follows:

Values of ΔL.	*Total Section.*	*Steel Section.*
$\Delta L_{13}=\Delta L'_0=$ 3.3 Ft.	$A_0=24.4$ Sq. Ft.	$A_{20}=.125$ Sq. Ft.
$\Delta L_{12}=\Delta L_1=19.5$	$A'_0=24.2$	$A'_{20}=.125$
$\Delta L_{11}=\Delta L_2=17.0$	$A_1=22.8$	$A_{21}=.1$
$\Delta L_{10}=\Delta L_3=14.8$	$A_2=19.2$	$A_{22}=.1$
$\Delta L_9=\Delta L_4=13.6$	$A_3=16.6$	$A_{23}=.08$
$\Delta L_8=\Delta L_5=12.9$	$A_4=14.3$	$A_{24}=.08$
$\Delta L_7=\Delta L_6=12.5$	$A_5=11.7$	$A_{25}=.06$
93.6 Ft.	$A_6=11.4$	$A_{26}=.06$

The moments of inertia, depths of rib, and the coördinates x, y, a and b, are shown below:

Moment of inertia.	*Depth of rib.*
$I_0=135.8$	$h_0=h_{14}=7.5$ ft.
$I_0'=132.2$	$h_0'=h_{13}=7.46$
$I_1=106.8$	$h_1=h_{12}=7.0$
$I_2=65.86$	$h_2=h_{11}=5.9$
$I_3=14.9$	$h_3=h_{10}=5.1$
$I_4=27.33$	$h_4=h_9=4.4$
$I_5=14.65$	$h_5=h_8=3.6$
$I_6=13.53$	$h_6=h_7=3.5$

$x_0'=$ 1.05 Ft.	$y_0'=$ 1.4 Ft.	$a_1=$ 2.1 Ft.	$b_1=$ 3.2 Ft.
$x_1=$ 8.	$y_1=11.20$	$a_2=14.35$	$b_2=18.4$
$x_2=20.4$	$y_2=24.4$	$a_3=26.6$	$b_3=29.8$
$x_3=32.4$	$y_3=34.$	$a_4=38.85$	$b_4=37.8$
$x_4=44.8$	$y_4=40.8$	$a_5=51.1$	$b_5=43.$
$x_5=57.2$	$y_5=45.$	$a_6=63.35$	$b_6=46.$
$x_6=69.6$	$y_6=46.8$	$a_7=75.6$	$b_7=47.$
$x_7=81.6$	$y_7=46.8$	$a_8=87.85$	$b_8=46.$
$x_8=94.$	$y_8=45.$	$a_9=100.1$	$b_9=43.$
$x_9=106.4$	$y_9=40.8$	$a_{10}=112.35$	$b_{10}=37.8$
$x_{10}=118.8$	$y_{10}=34.$	$a_{11}=124.6$	$b_{11}=29.8$
$x_{11}=130.8$	$y_{11}=24.4$	$a_{12}=136.85$	$b_{12}=18.4$
$x_{12}=143.2$	$y_{12}=11.2$	$a_{13}=149.1$	$b_{13}=3.2$
$x_{13}=150.15$	$y_{13}=1.4$		
$x_{14}=151.2$ (span)			

The results of the successive applications of a single force P to each point of loading throughout the entire rib will next be found.

The quantity M^1 given by Eq. 6a will now take the form:

$$M^1 = P \cdot (x - a_1) \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots 67$$

Consequently the terms in Eqs. 40, 41 and 44 in which M^1 appears will become:

$$\sum_0^l \frac{M' y \Delta L}{I} = P\left[\sum_{a_1}^l \frac{xy\Delta L}{I} - a_1 \sum_{a_1}^l \frac{y \Delta L}{I}\right] \ldots\ldots 68$$

$$\sum_0^l \frac{M' \Delta L}{I} = P\left[\sum_{a_1}^l \frac{x \Delta L}{I} - a_1 \sum_{a_1}^l \frac{\Delta L}{I}\right] \ldots\ldots\ldots 69$$

$$\sum_0^l \frac{M' x \Delta L}{I} = P\left[\sum_{a_1}^l \frac{x^2 \Delta L}{I} - a_1 \sum_{a_1}^l \frac{x \Delta L}{I}\right] \ldots\ldots 70$$

The following tabulation illustrates the application of the preceding data to Eqs. 38, 39, 40, 41, 42, 43 and 44 for the determination of the quantities a, b, c, a', b', c', a'', b'' and c''.

a_1	$\sum_a^l \frac{x\Delta L}{I}$	$-a_1 \sum_{a_1}^l \frac{\Delta L}{I}$	$+\sum_0^l \frac{M'\Delta L}{I}$
2.1 Ft.	474.28	— 13.05	+ 461.23
14.35	472.81	— 86.58	+ 386.23
26.50	467.55	— 153.62	+ 313.93
38.85	456.11	— 210.64	+ 245.47
51.10	431.33	— 251.64	+ 179.64
63.35	380.99	— 256.22	+ 123.77
75.60	316.68	— 235.91	+ 80.77
87.85	241.28	— 192.52	+ 48.76
100.10	158.56	— 131.28	+ 27.28
112.35	105.62	— 91.44	+ 14.18
124.60	63.66	— 57.40	+ 6.26
136.85	29.90	— 27.72	+ 2.18
149.10	3.75	— 3.72	+ 0.03

The following values, which do not involve the load P, at once result:

$a = -1.78416$ (from Eq. 38).
$b = 0.0236$ (" " 39).
$a' = -1.90966$ (" " 41).
$b' = 0.0256$ (" " 41).
$a'' = -2.26366$ (" " 42).
$b'' = b' = 0.0256$ (" " 43).

The quantities c, c' and c'' (Eqs. 40, 41 and 44) will each have a

separate value for each point of application of P. If P be taken as unity, these values will be:

a_1	c	c^1	c^{11}
2.1 Ft.	1.75166	1.86655	2.21038
14.35	1.46096	1.56303	1.89968
26.60	1.17517	1.27044	1.59116
38.85	0.90056	0.9934	1.28823
51.10	0.64495	0.72715	0.99718
63.35	0.42454	0.50088	0.73052
75.60	0.25195	0.32687	0.50095
87.85	0.13219	0.19733	0.31813
100.10	0.06084	0.1104	0.18656
112.35	0.02459	0.05739	0.09914
124.60	0.00745	0.02533	0.04268
136.85	0.00151	0.00882	0.01170
149.10	−0.0000277	0.00012	0.000212

The negative and exceedingly small value of c for $a_1 = 149.1$ is obviously a small error in the application of the formulae for that quantity due chiefly to lack of exactness in the procedure. The value should be but little more than zero, and positive.

Introducing the above numerical values in Eqs. 44a, 44b, and 32, there will result:

$$M_0 = 177.26\,(c'' - c') - 500\,(c' - c) \quad \ldots\ldots\ldots\ldots 71$$

$$V = 2.82486\,(c'' - c') \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots 72$$

$$H = 1.78416\,V - 0.0236\,M_0 - c \quad \ldots\ldots\ldots\ldots 73$$

These are the equations of the influence lines for M_0, V and H, which are shown graphically on Plate 1.

The use of the numerical values of c, c' and c'' in Eqs. 71, 72 and 73 will give the following values for M_0, V and H:

a_1	M_0	V	H
2.10 ft.	+ 3.502 lb.	.971 lb.	
14.35	+ 8.626	.951	
26.60	+ 9.213	.906	
38.85	+ 8.700	.833	
51.10	+ 6.763	.763	
63.35	+ 2,534	.649	
75.60	— 6.604	.492	0.78 lb.
87.85	— 11,158	.341	0.735
100.10	— 11.291	.215	0.59
112.35	— 9.000	.118	0.4
124.60	— 5.865	.049	0.218
136.85	— 2.500	.008	0.072
149.10	— 0.044	.00026	0.0015

These values are for one pound (or other unit load) placed at the points on the rib located by the horizontal coördinates a_1.

The end moment M_0 and the vertical and horizontal forces V and H for any load at any point, can at once be found by multiplying the above unit value of the same quantity (taken from the preceding table or scaled from Plate 1) by that load.

The general values for the normal or shearing force (N or S) or for the bending moment M at any point given by the equations 4, 5 and 6, may now be determined by simply adapting those equations to vertical loading as indicated by Eq. 44c and in the paragraph preceding that equation. These operations will enable the influence lines for M, N and S to be drawn. The complete conditions of stress in the entire rib, or its complete design, may then be treated as set forth in Art. 5.

A STUDY OF MECHANICAL CONVEYORS.—I.*

BY LINCOLN DeG. MOSS.

The Ton-Mile as a Basis of Comparison.— Evolution of Mechanical Transport.— Efficiencies of Various Methods.— Vertical Hoisting by Man and by Horse Power.— Chain-Bucket Conveyors: Methods of Discharge; Types of Chain; Boots; Power Consumed.— Gravity Pivoted Bucket Conveyors: Hunt System; McCaslin System; Dodge System; Dumping Devices; Computation of Power Consumed.— Flight Conveyors: Types; Construction.— Dodge Coal Storage.— Reciprocating Conveyors.

The mechanical efficiency of transporting devices should be compared on the basis of the power used to move one ton one mile. Commercial efficiency is best measured by the total cost per ton-mile, as in railway and marine transportation. It does not follow that a system which consumes the least power per ton-mile is the most desirable: we must also consider suitability, first cost, maintenance and fixed charges. Costs based on the tons delivered per hour alone, without stating length of haul, are misleading, though often quoted. For an extreme example, a certain screw conveyor delivers 50 tons of grain per hour and consumes 18 h. p., delivering the grain 100 ft. from the point of loading. The same expenditure of power would deliver a like tonnage per hour more than 100 times as far by an aerial tramway. Hence the necessity for a rational standard such as the ton-mile for a basis of comparison of gross cost and power consumption. Elevator service is frequently compared on the basis of the car-mileage, which is clearly erroneous, for the car is a variable in weight and paying capacity, whereas the ton, the mile, and the horsepower are unvarying standards. The cost per ton, and horsepower per ton are commonly used, but a little reflection shows their fallibility as true standards of measurement. Grades can be equated, as in railway work; or the energy of lifting can be taken out or added bodily; or we may say that the work of lifting (W) is equal to W $\times$ x ft. of horizontal conveyor. No particular type of conveyor is universal; each has its peculiar advantages and limitations.

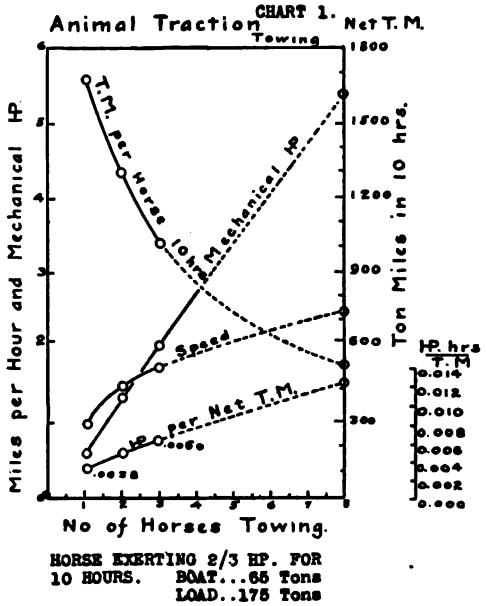
* Extracts from the first three lectures on "Elevating and Conveying Machinery," delivered before the Department of Mechanical Engineering, Columbia University.

Glancing backward over the short history of man and his early efforts in transportation, we find mention of the spade and wheel early in the Old Testament. The wheel and pulley, ropes and rollers appear on the carvings of the Assyrians and Egyptians. Buckets were used on the rims of the Persian wheel and noria. It is notable that all these ancient methods survive. Archimedes' screw still pumps on the banks of the Nile; the long sweeps of the shadoof, ancient prototype of the giant Hulett ore unloader, still rise and fall in Egypt. In the West Indies, and many Old World seaports, living chains of men and women still carry coal in baskets to and from steamers. Man soon learned to remove this 50 to 60 lb. load from his head and place it over a wheel which bore the major part of the load. This expedient permitted him to triple the load with no loss of speed, and with less fatigue. By putting the load in a cart with two large wheels, giving the cart body a proper profile, as seen in the charging car, and providing a fairly smooth runway, the load was increased to 500 lb. The next step was to place the load on four wheels running on a metal track. This made it possible for a man to push a net load of one ton at a speed of 100 feet per minute.

With the advent of machinery, the bookkeeper appeared and began to charge up the cost of repairs and the interest which his employer must pay for the loan which enabled him to purchase the plant. A sinking fund was also established to repay the loan and purchase a new plant at some future time. Simple as these principles are, contractors and corporations are failing, in these days, for not complying with them. Therefore the true cost to consider is the gross cost per ton-mile, which gives due value to power consumed, wages, repairs, depreciation, interest on capital, and sinking fund, and allows for the distance through which the work is accomplished. As wages and power decrease, with more efficient machinery, fixed charges increase, as a general rule.

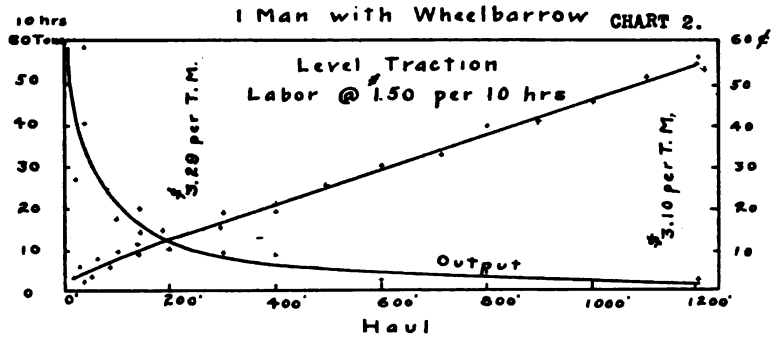
Man soon employed the horse and a few other beasts of burden to aid him. A good horse will develop ⅘ h. p. for 10 hours, at low speed; an able bodied man, 1-10 to 1-11 h. p. for 10 hours. Man, in proportion to his weight, is stronger than a horse or an elephant. A strong man will develop, in 10 hours, about 1 h.p.-hour, for which we must pay $1.50 usually. The extreme cost of his fuel

$200 and upwards per ton, largely accounts for this. With all our mechanical triumphs, the horse towing a canal boat remains today mechanically the most efficient transporter, although the vessels drawn are heavy and crude, designed with no reference to the "coefficient of fineness," and travel through a shallow channel. Nevertheless, a horse can move one of these crude carriers weighing 65 tons, with 175 tons of paying load aboard, 10 miles in 10 hours, giving 1750 net ton-miles with an expenditure of 6⅔ h.p.-



hours, or 0.0038 h.p.-hour per ton-mile. The nearest approach to this is found in ocean towing, which requires 3¼ times as much power per ton-mile. Large lake steamers and ocean freighters follow in efficiency, with coefficients of 0.015 and 0.02 respectively, while large freight trains give coefficients of 0.03 to 0.05. Sir William White says that 1000 ton-miles at sea cost for fuel 2½ to 3 pence; 3 shillings' worth of coal will drive a ton from London to Australia, in general at a cost of 1-35 that of railway transportation per ton-mile. In the department of mechanical conveyors, we find the Hunt gravity railway consumes the least power, 0.15 h.p.-hour, per net ton-mile, which is 40 times as much as that developed by

the towing horse per net ton-mile. The mechanical efficiency decreases steadily throughout this field, until we reach the useful and often desirable screw conveyor, which requires 5000 times as much power per net ton-mile as is exerted by the towing horse.



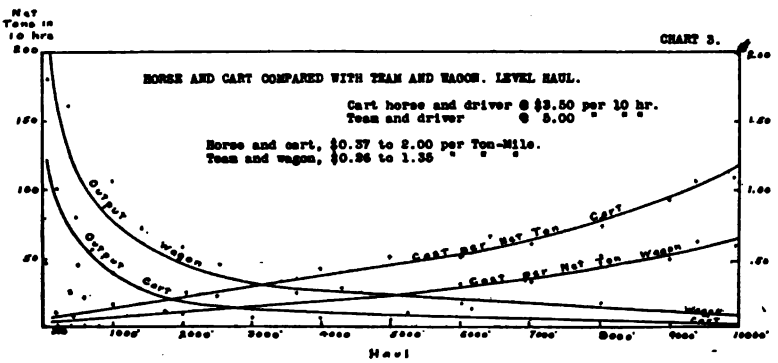


CHART 3.
HORSE AND CART COMPARED WITH TEAM AND WAGON. LEVEL HAUL.
Cart horse and driver @ $3.50 per 10 hr.
Team and driver @ 5.00
Horse and cart, $0.37 to 2.00 per Ton-Mile.
Team and wagon, $0.26 to 1.35
Output Wagon
Output Cart
Cost per Net Ton Cart
Cost per Net Ton Wagon
Haul

Chart 2 shows the cost of moving materials by wheelbarrows on level runs of planks to various distances. Chart 3 shows similar data for hauling by carts and wagons. Chart 4 compares the cost and the work done by men and horses moving cars on T-rail tracks. Chart 5 shows the gross cost per ton and per ton-mile by these methods.

Chart 6 shows the evolution to small trains of cars drawn by locomotives. The data refer to the performance of a train of 10 dump cars, each holding 1½ tons, drawn by a locomotive of the tank pattern, weighing 10 tons, all on 4 drivers. For comparison, the same chart shows the output and the cost of working on large enterprises, with two trains, each having a locomotive and 25 modern

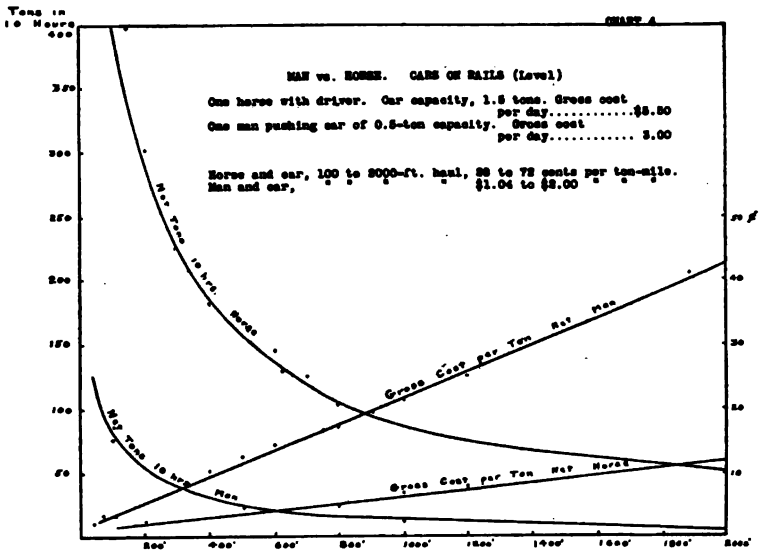


CHART 4.
MAN vs. HORSE. CARS ON RAILS (Level)
One horse with driver. Car capacity, 1.5 tons. Gross cost per day.....$5.50
One man pushing car of 0.5-ton capacity. Gross cost per day..... 3.00
Horse and car, 100 to 8000-ft. haul, 36 to 72 cents per ton-mile.
Man and car, " " " " $1.04 to $2.00 " " "
Net Tons in 10 Hours
Net Tons in 10 hrs. Horse
Net Tons in 10 hrs. Man
Gross Cost per Ton Net Man
Gross Cost per Ton Net Horse

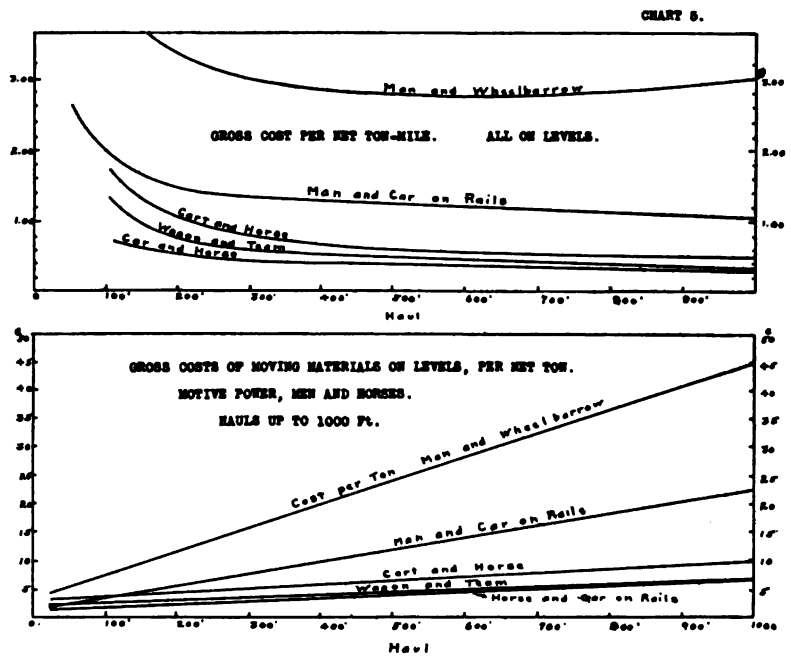


CHART 5.
Man and Wheelbarrow
GROSS COST PER NET TON-MILE. ALL ON LEVELS.
Man and Car on Rails
Cart and Horse
Wagon and Team
Car and Horse
Haul
GROSS COSTS OF MOVING MATERIALS ON LEVELS, PER NET TON.
MOTIVE POWER, MEN AND HORSES.
HAULS UP TO 1000 Ft.
Cost per Ton
Man and Wheelbarrow
Man and Car on Rails
Cart and Horse
Wagon and Team
Horse and Car on Rails
Haul

self-dumping cars of 20-cu. yd. capacity, the source of supply being a first-class steam shovel with a 4-cu. yd. dipper, kept constantly at work, and capable of delivering 5000 tons in 10 hours. The dotted lines on this diagram indicate the rapid increase in cost per ton by animal traction. Do not expect to excel the performances shown on these charts except in the most favorable cases; and if there are grades to be overcome, make due allowance. In 10 hours, a horse should not be worked beyond 9,000,000 ft. lb. gross, at the traces, nor a man beyond 1,500,000 ft. lb. at his hands.

Horizontal conveying merges into inclined conveying, when the work of lifting becomes a factor. It is then a short transition to systems in which there is no horizontal work, all the energy being expended in vertical lifting, plus some inevitable frictional losses. In some inclined systems, the loads are lowered as well as moved horizontally.

Chart 7 shows the work performed and the cost of vertical hoisting by two men on a windlass, compared with a treadmill or a capstan. Man works more efficiently on the latter machines, but the compactness and handiness of the windlass make it a general favorite. The capstan has a vertical barrel, as distinguished from the windlass, which has a horizontal barrel. The capstan is propelled by bars inserted in sockets in the capstan head, permitting the men to exert a uniform torque, aided by the push of their legs, in breasting the bars. Treadmills, still used in the Old World, are as much as 20 ft. diameter and are as efficient as the capstan. The same chart shows the output and the cost of hoisting by a horse, first, hauling on a single whip, or a two-fold purchase, and second, hoisting by a capstan, or horizontal drum actuated by a beam, which the horse draws as he walks in a circular path of about 20 ft. radius. The drum has a friction clutch and brake for lowering the load, which is done by the driver while the horse stands and rests.

In future references we will designate as "continuous conveyors" those consisting of endless chains with flights or buckets, or endless bands which move continuously, vertically, obliquely, horizontally, or in various combinations of directions. By "intermittent conveyors" we mean those which start from rest, accelerate the load, and stop when the load is ready to be dropped, such as a hoisting engine operating a skip or platform elevator. Between these two

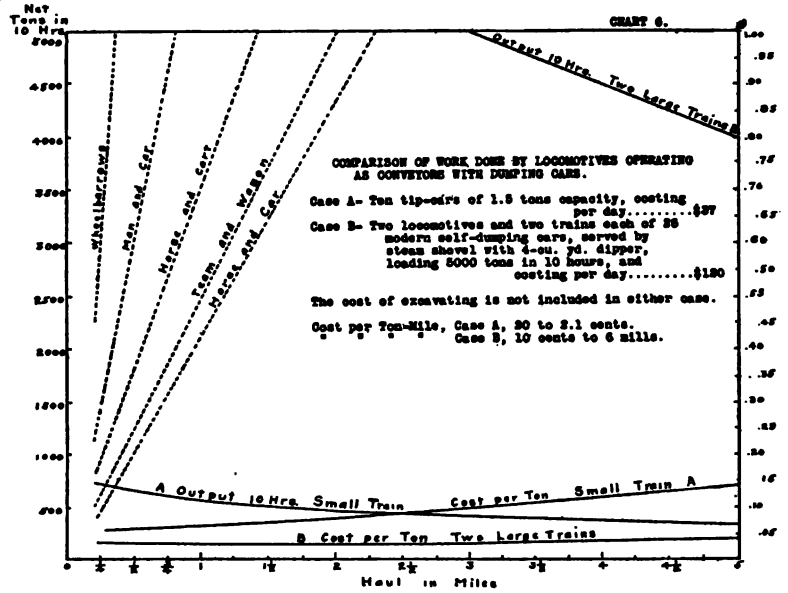
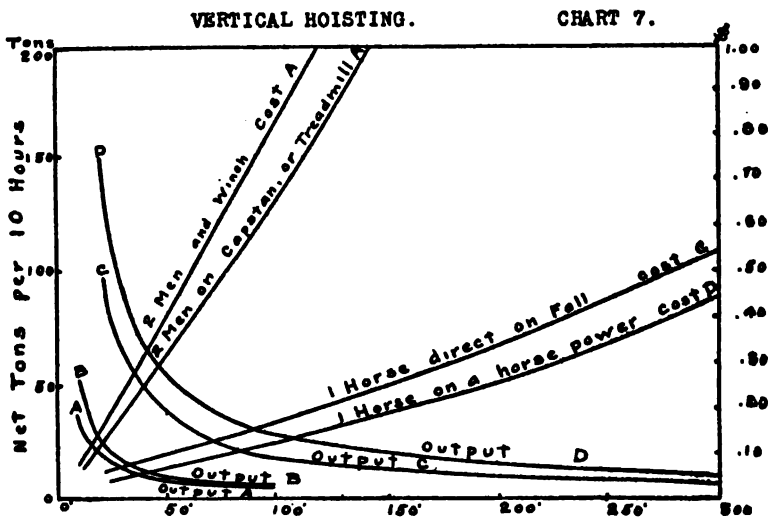


CHART 6.
Net Tons in 10 Hrs.
Output 10 Hrs. Two Large Trains
Wheelbarrows
Men and Car
Horse and Car
Team and Wagon
Horse and Car
COMPARISON OF WORK DONE BY LOCOMOTIVES OPERATING AS CONVEYORS WITH DUMPING CARS.
Case A- Ten tip-cars of 1.5 tons capacity, costing per day.....$37
Case B- Two locomotives and two trains each of 26 modern self-dumping cars, served by steam shovel with 4-cu. yd. dipper, loading 5000 tons in 10 hours, and costing per day.....$120
The cost of excavating is not included in either case.
Cost per Ton-Mile, Case A, 20 to 2.1 cents.
" " " " Case B, 10 cents to 6 mills.
A Output 10 Hrs. Small Train
Cost per Ton Small Train A
B Cost per Ton Two Large Trains
Haul in Miles



A- Two men on windlass, cranks at 90°, radius 18 in.
B- Two men on a capstan or a treadmill.
C- One horse pulling on a single whip, or a two-fold purchase.
D- One horse pulling on a "horse-power".

classes are the "semi-continuous," such as endless aerial tramways, or endless cable railways.

Chain-Bucket Conveyors.

Bucket conveyors have a wide range of usefulness for vertical, inclined, and horizontal conveying. Buckets range in size from 3x3x3 in. to the huge buckets of seagoing dredges, which hold over 1 cu. yd. and, at a speed of 16 buckets per minute, have dredged 850 net cu. yd. per hour from a depth of 32 ft. We may classify chain-bucket elevating conveyors, with reference to their mode of discharging the buckets, into four general types:

(A) Centrifugal Discharge. The material leaves the buckets at the head sprocket, clearing the elevator well by centrifugal momentum. This method is not suitable for sticky materials, and will cause dust and breakage of soft materials, which may be objectionable in some cases.

(B) Perfect Discharge. The descending chains leave the head wheels and are bent inward by snub sprockets to invert the buckets over a chute, giving a clean and gentle delivery.

(C) Gravity Discharge. The loaded buckets rise to the head sprockets and then pass horizontally over a chute or a bin, the material falling out of the buckets by gravity. V-shaped buckets are required for this type. By making the horizontal run extend over a bunker, and providing a trough with gates, this type of elevating conveyor has an enlarged field of usefulness by the added function of a horizontal conveyor. The chain line, after leaving the trough, may descend and then return horizontally in a lower trough for loading. The cheapness of this form of conveyor for dual purposes is an offset to the larger power required to operate it, as compared with the more refined carriers which do not use fixed troughs.

(D) Continuous Type. The buckets come together or overlap slightly. The speed is low. The buckets, when inverted over the head wheels, discharge their contents over the sloping bottoms of the inverted buckets, which serve as deflecting chutes. This type of conveyor is suitable for very large quantities, and is generally used for coarse and heavy materials.

The shape of the buckets varies to suit the material, and the man-

ner of discharging. Sheet steel is most generally used, but for hot, abrading and corrosive materials, annealed malleable iron is more durable. The edges of malleable iron buckets are generally thickened to increase durability. Buckets may be perforated for draining wet materials in transit.

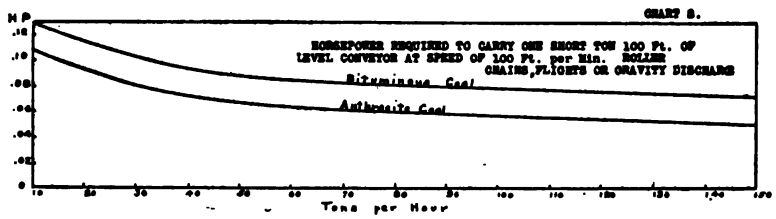
Speeds of 165 to 400 ft. per min. are used for centrifugal discharge elevators having head wheels from 15 to 48 in. diameter, the lower speeds being used on the smaller wheels. Perfect discharge elevators are run at speeds of 80 to 200 ft. per min. Gravity discharge elevators operate at 60 to 200 ft. per min. as do also continuous elevators. Rubber and canvas belts are much used for grain elevators, to some extent for mud and earth, and occasionally for coal and stone. While free from corrosion, they must be arranged so as to prevent gritty materials from entering between the belt and the foot pulley, which would cut the belt.

Chains are made in a wide variety of forms. Single strand chains are generally used for centrifugal discharge elevators, but cannot be used for the perfect discharge type. Single chains on the backs of the buckets encounter the material at the loading boot. This is often permissible, but may be avoidable in gritty material by using two strands bolted to the sides of the buckets. If the material is very sharp, like glass sand, the chains should be set farther out on spacer rods. The Ewart detachable, malleable-iron chains are widely used. For very heavy service they are inferior to bouched and special chains, such as the Ley, Dodge, Scranton, and others. Case-hardened pins and steel and bronze bouchings are used occasionally. For extremely heavy service, forged chains, frequently with upset ends and steel bouchings are employed. Roller chains are best with gravity discharge conveyors, the rollers diminishing the friction at the horizontal portion of the run.

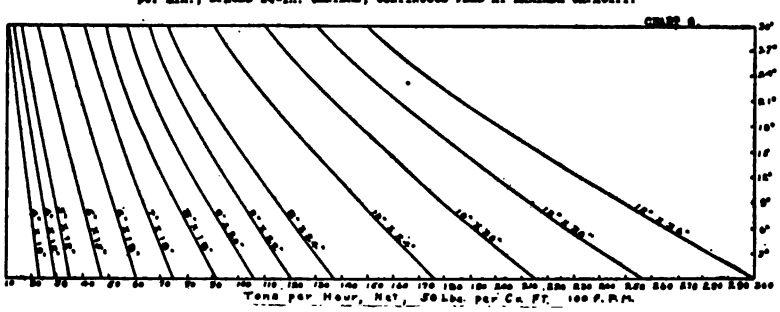
It is frequently found that the bearing pressure on chain pins is well beyond those pressures per square inch which are associated in our minds with good lubrication. This is especially true of low-speed chains. The horsepower may be small, but the low speed of transmission causes a surprisingly larger tension in the chains and heavy journal pressure. There is a small angular motion of the links on the pins when the chain passes over a sprocket wheel. We need not fear to use high pressures, provided they do not exceed the

pressure which will squeeze out a heavy-bodied lubricant. Cases have been known where bearing pressures exceeded the bearing value of the metal, elongating the holes in the links and reducing the diameter of the pins.

The strength of the chains and the bearing pressures on their articulations govern the practicable height of lift. The stress in the chains should not exceed $\frac{1}{8}$ the ultimate strength; more margin is desirable in high and important elevators, in which a breakage of the chains would reduce the machinery to a heap of junk. For filling the buckets, a box enveloping the lower loop of the chains,



CARRYING CAPACITIES OF FLIGHT CONVEYORS AT VARIOUS ANGLES. SPEED, 100 ft. per min.; SPACED 24-in. CENTERS; CONTINUOUS FEED AT MAXIMUM CAPACITY.



and containing the sprockets, shaft, and journals is built, preferably with cast-iron sides, connected by a curved sheet-steel bottom, the sides being cross connected by through bolts. The feed should be to the ascending buckets. When possible, the journals should have screw take-ups. A wear of 0.01 in. on 400 four-inch links will increase the length of chain by 4 in. When the strength exceeds the range of the take-up, links are detached and the journals set back again. Local conditions sometimes compel us to mount the head shaft journals on take-up screws, with considerable weight to handle when adjusting. For gravity discharge elevators, it is better to place the take-up on the top horizontal run at its far end.

The net work of lifting is quickly computed by the handy formula: Tons per hour (net) × vertical height of elevator ÷ 990 = net horsepower. A divisor of 1000 may be used, as the error is small. We have found elevators which gave 0.75 efficiency at the head shaft; ⅔ to ⅘ is more common. For an ordinary motor-driven elevator, the motor horsepower at the brushes, (Ampères×Volts ÷ 746) will generally be 0.003 × tons per hr. × ft. lift. This allows for 0.80 efficiency of motor; 0.70 efficiency of drive; and 0.60 efficiency of elevator. For first-class elevators, the multiplier may fall to 0.002, the efficiency of the motor being 0.90; of the drive, 0.85; and of the elevator, 0.67. It is advisable to calculate capacity with buckets ⅘ full, but calculate the power for the buckets fully loaded.

The bucket elevator is in balance, and has none but paying load to lift. Acceleration enters only when starting, which can generally be done with no load in the buckets. In that case, a shunt-wound motor will answer. If local conditions compel us to start up with load, a compound winding must be used, giving a good starting torque and accelerating gradually, with good speed regulation. A series motor is out of place in such service, lacking speed regulation, starting too rapidly, and being apt to run away and burst its armature if the chains part. Long, easy running, motor-driven elevators with heavy loads may back down by gravity if the power is cut off. To prevent this, a mechanical brake can be applied to the head shaft, or a solenoid brake can be applied to the motor shaft, which will set automatically when the circuit is broken. The latter method is safe if the gearing between motor and head shaft remains intact.

Sprocket wheels are usually made of grey cast iron, but for heavy work are cast steel, or have either chilled teeth or removable wearing faces. Traction wheels without teeth are used for single-strand elevators which carry very gritty materials.

Gravity Pivoted Bucket Conveyors.

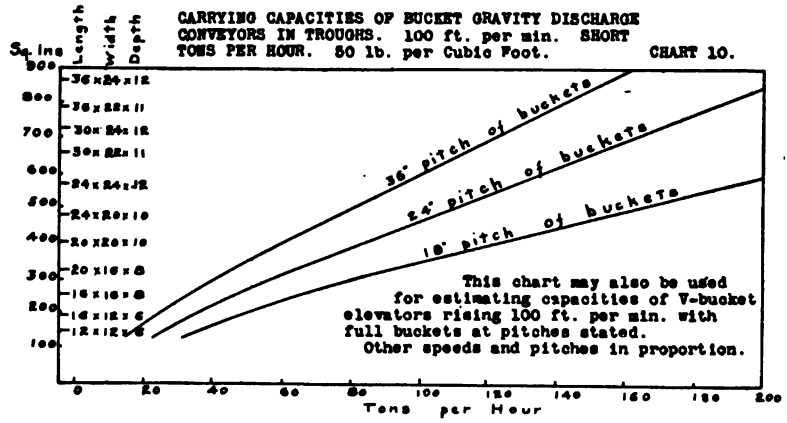
The buckets are pivoted to supporting roller chains, so that they may remain plumb regardless of the changing positions of the chains, which may move horizontally, vertically, obliquely or in combination of all three directions. In its original form, as introduced by the C. W. Hunt Company, in 1885, the buckets swung

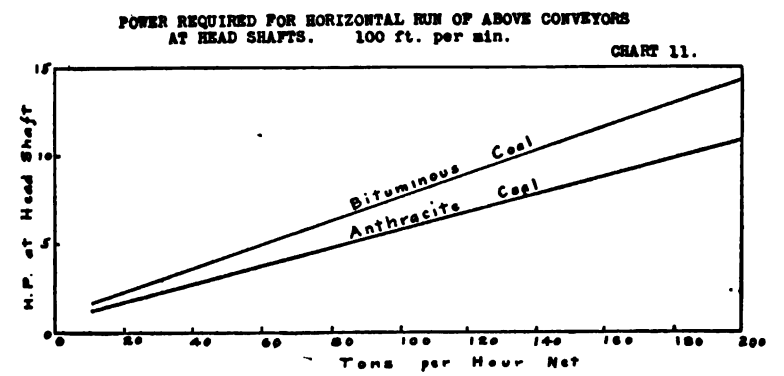
clear of each other and were filled by three methods. (1) A filler, sometimes movable on a track, carried two endless chains driven by the conveyor chains. The endless chains carried cast-iron hoppers moving in step, and coinciding with the mouths of the conveyor buckets, thus forming beveled bridges over the spaces between the buckets. This method is still preferred by the C. W. Hunt Company. (2) In some cases, a rotary filler is used by that company. It has a cylindrical revolving cutter which measures out one bucket load at each half revolution, and drops it into a conveyor bucket. The rotary cutter derives power from the conveyor chains and has bi-lobular gears, commonly called elliptic gears, which cause the cutting to take place at low speed, with a quick positive discharge. This filler is not used for coarse materials. (3) Star filler, which is simply a pair of small circular heads which revolve on a shaft, having bridge sheets to cover the spaces between buckets, and open spaces between the sheets which coincide with the bucket mouths. The filler is driven by the conveyor chains so as to revolve in step with the buckets. The C. W. Hunt Company prefers to use revolving wheels at the upper corners of the more or less rectangular path of the conveyor, and at the lower rising corner when the chain pull is heavy. These wheels have radii of 2 to 3 ft. Cast-iron curves with strap-steel faces were originally used, but the wheels reduce the friction of turning corners. It may be demonstrated mathematically that, with a given chain pull on a 90° turn, a wheel of 3-ft. radius consumes half the power of a fixed curve of 6-ft. radius, which in ordinary practice would save 75 lb. in chain pull. With larger radii of fixed curve, this disadvantage is reduced. The revolving corner wheels require more clearance than fixed curves.

The Hunt conveyor uses steel roller chains of flexible construction, which permits the chains to be twisted in order to change the horizontal direction of the conveyor. A horizontal turn of 90° can be made in a vertical rise of 25 to 28 ft. The Hunt driver is unique. In place of sprocket wheels, a number of pawl arms are arranged to push on the chain studs. These pawls revolve with a driven shaft which carries two heart cams; these enter the pawls and withdraw them so that when one pawl has nearly completed its stroke the next one takes hold. This automatically compensates

for the increase of lost motion in the chains as they lengthen by wear. The largest Hunt conveyor handles 250 tons of phosphate rock per hour with buckets 42 in. wide.

The George W. Caslin conveyors, introduced about 1889, have maintained their original characteristics such as the overlapping lips, malleable-iron buckets, chains with reinforced ends, fixed


CARRYING CAPACITIES OF BUCKET GRAVITY DISCHARGE CONVEYORS IN TROUGHS. 100 ft. per min. SHORT TONS PER HOUR. 50 lb. per Cubic Foot.
CHART 10.
Length
Width
Depth
Cu. Ins
36" pitch of buckets
24" pitch of buckets
18" pitch of buckets
This chart may also be used for estimating capacities of V-bucket elevators rising 100 ft. per min. with full buckets at pitches stated. Other speeds and pitches in proportion.
Tons per Hour

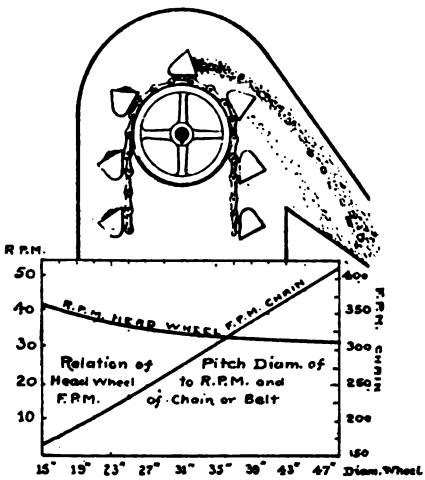

POWER REQUIRED FOR HORIZONTAL RUN OF ABOVE CONVEYORS AT HEAD SHAFTS. 100 ft. per min.
CHART 11.
H.P. at Head Shaft
Bituminous Coal
Anthracite Coal
Tons per Hour Net

curves of long radii, and no fillers. Using overlapping lips, and four changes of direction, it becomes necessary to dip the emptied buckets on the descending run in order to place the forward lip of a bucket under the after lip of the preceding bucket, so that the loaded buckets may enter the first lift without breaking off the lips. This is accomplished by curved guides which engage the dumping cams or the buckets and dip them to the proper position. With this construction, no filler is needed, other than a pair of skirt

boards, as there can be no leakage between the buckets. McCaslin conveyors have been used for moving hot cement clinker, and it is interesting to note that the effect of heat on the malleable-iron buckets does not render them brittle, but more malleable.

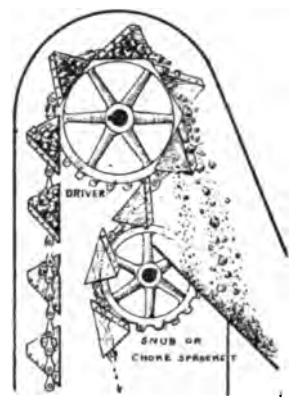
CHART 12

Centrifugal Discharge.



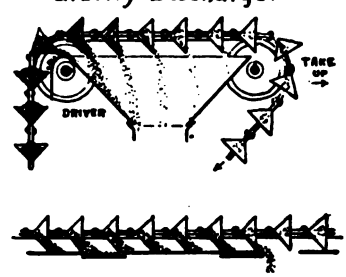
Used with one or two strands of chain or a belt. Single chain may run on a plain traction wheel if the material is very gritty.

Perfect Discharge.



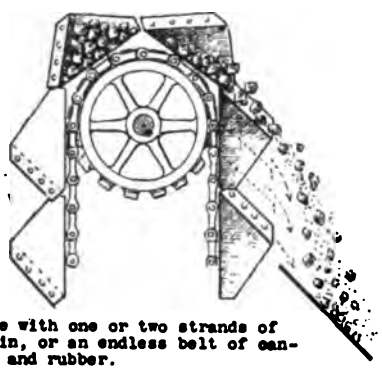
Speed of chain, 80 to 100 ft. per min. Requires two strands of chain.

Gravity Discharge.



Frequently arranged with long level runs to convey material horizontally on upper or lower run, in addition to elevating. Roller chains should be used.

Continuous Discharge.



Made with one or two strands of chain, or an endless belt of canvas and rubber.

L.B.&M.

The Dodge pivoted-bucket conveyor avoids the necessity for a filler, and for overlapping buckets, by placing the buckets quite close together, yet never touching, and catching the fine spill between buckets in small "retriever" buckets; these are suspended

from the chains so as to be directly under the narrow gaps between main buckets, and are inverted on the run over the bunkers. The Jeffrey Manufacturing Company builds this type of conveyor in two forms. (1) With overlapping lips and no filler. (2) With "close working" buckets having planed edges, and fed by a clever cut-off filler which is essentially a cut-off gate and arm actuated by the conveyor chain.

Conveyors of this type have been widely used in boiler houses, putting in coal and taking out ashes by the same machine, and delivering both to overhead bunkers. When furnace ash is to be carried, care should be taken to prevent corrosion of the chains, and to exclude the grit from them and from the track. Ashes and water usually give a strong alkaline reaction; they sometimes carry sulphur, and are strongly corrosive. If sheet-steel buckets are used for carrying wet ashes, they should be cleaned after such use, to prevent the formation of a firm, pasty layer of damp ashes in the bottom and lips of the buckets. Cast-iron buckets, for such service, are much more durable, resisting corrosion and mechanical wear, but breaking where sheet-steel buckets would bend. Neglect in the care of such apparatus has, in a number of cases, caused their removal, and the substitution of other devices. It is remarkable how much thought and care are expended in some power stations to install the most efficient motive power, and operate it at high efficiency, while permitting an ill kept, neglected conveyor in the boiler room. Conveyors generally are wasteful in power consumption. Fortunately the power required, in comparison with the total output of a station, is small. Good care of conveying machinery is of more moment in prolonging life and preventing breakdowns.

Dumping is effected by trippers with inclined planes, which engage dumping cams on the buckets. The Jeffrey buckets have loose wheels in place of cams, to reduce friction. The dumpers can be stationary at dumping points, and have folding arms which can be withdrawn to permit the buckets to pass without emptying. They may also be mounted on a carriage running on rails, and moved by rope and windlass. A number of McCaslin machines have been arranged to dump alternately to different bins, for cement handling, by using cams of different widths, and, if further

alteration is desired, using cams of varying width on two sides of the buckets.

Capacity ranges from 30 to 150 tons per hour for boiler house work. Speed ranges between 34 and 50 ft. per min. and is limited largely by the blow of the bucket cams on the dumpers. Power is estimated by widely varying methods. One of the shortest, and one which has always proved reliable, ascertains the motor horsepower by finding the size of conveyor required, the necessary speed, and the weight of buckets, chains and coal combined per linear foot of conveyor. Then compute the total weight of horizontal loaded conveyor plus total weight of horizontal unloaded conveyor, and take ⅛ of the result. Add the weight of coal contained in buckets of the vertical run, cancel the rising part of the vertical conveyor with the descending portion; multiply by speed of chains in ft. per min. and divide by 33,000 foot-pounds. Other methods are more elaborate and analytical. We suggest the following formula, which saves time and labor of working out the size, weight and speed of an adequate conveyor, yet gives due value to the variations in size and weight for varying capacities:

$$\text{H. P.} = \frac{T \times (R + \frac{1}{10} H + \frac{1}{20} h)}{1000} \times C, \text{ in which:}$$

T=net tons per hour.
R=verical rise, in feet.
H=horizontal length of loaded conveyor, in feet.
h=horizontal length of empty conveyor, in feet.
C=a constant, as below.
H.P.=input at motor.

VALUES OF C IN ABOVE FORMULA

	Tons per Hour	Vertical lift, Ft.	C
Small Conveyors	30 — 40	40 — 60	2.2
		60 — 90	2.4
		90 — 120	2.6
Medium Conveyors	45 — 60	40 — 60	2.0
		60 — 90	2.2
		90 — 120	2.4
Large Conveyors	70 — 150	40 — 60	1.8
		60 — 90	2.0
		90 — 120	2.2

The answer is in terms of motor horsepower, and allows for motor losses, friction at curves, and drive and traction losses.

Flight Conveyors.

The flight conveyor is related to the bucket type, the flight being a bucket without sides. It is used for inclined and horizontal conveying. It is not desirable to run them on inclines steeper than 30°. The carrying capacity decreases with the pitch, as the quantity of material pushed ahead of each flight decreases with the increase of the angle. However, when carrying material like mine-run coal, containing many large irregular lumps, this disadvantage is less marked, as the lumps bridge and hold the fine material from sliding back. See charts 8 and 9 for capacities and power. This type of conveyor is relatively cheap and widely useful. Inasmuch as the materials are dragged through a trough, more friction is encountered than in the more refined types of bucket or apron carriers on wheels. Flight conveyors are made in three forms, classified as follows:

(A) The simplest form is the plain flight or scraper. The flights, or palms, rest upon the trough and are dragged through it by one or two chains. This form is the cheapest in first cost, but requires the most power and is prone to squeal when at work.

(B) Suspended flights. Each flight is bolted to a hanger on each side, with a cast-iron wearing block which rides on a greased guide bar on each side of the trough. The shoes are sometimes bored to slip easily over a round pin on the hanger, the blocks being two to three times as long as they are high, and with rounded ends, so that if they strike an obstruction the shoe will capsize and ride over it. The flights are suspended clear of the trough by a clearance depending on the material to be handled. Some operators make this clearance generous, so as to maintain a thin covering This type, with well greased tracks, requires about $\frac{1}{8}$ less power for a given material than type A.

(C) Roller flights. Wheels, preferably cored out and packed with oil-soaked fibre, are substituted for the wearing blocks. This refinement adds to the first cost, but decreases the power required for driving to $\frac{3}{4}$ of that needed for type A.

Flights are generally made of sheet steel, in a wide variety of sections. Some are curved, and thus bear a small portion of the

vertical load. When malleable iron is used for flights, to increase durability, the edges should be thickened. Plain drag flights require inclined sides. Rectangular flights are often used with types B and C, giving greater cross-section for carrying, and a more economical form of trough to construct. For long hauls, with moderate loads, the flights may be circular plates running in a V trough, using a chain or wire rope. These are much used for moving timbers and logs.

The requisite structural strength is frequently obtained by the use of timber stringers with a plank bed upon which the sheet-steel sectional trough is bolted. Hard tank plates will outwear high-grade steel sheets. Cast iron is too expensive for general use, but acquires a high polish and is an ideal wearing material. It is liable to warp in cooling and present an irregular section. All-steel construction is becoming more popular, and for permanent installations is cheaper in the end. Flights are usually spaced 18 or 24 in. centres, rarely less than 16 in. nor more than 36 inches.

Chart 8 shows the power required at head shafts to move 1 ton 100 ft. at the usual speed of 100 ft. per min. in level flight conveyors with roller chains, and those portions of gravity discharge conveyors which convey horizontally. Two curves are given, one for anthracite and one for bituminous coal. If a grade is to be overcome, add the net work of lifting, by formula, tons per hour $\times$ height in feet $\div$ 990, and allow for the additional friction due to the increased pull. The vertical portion of a gravity discharge elevator-conveyor is to be treated as a plain elevator, and added to the result.

A very useful application of continuous flight conveyors is seen in the Dodge system for storing large volumes of anthracite coal, which must be deposited gently to avoid breakage. Such coal may be piled high as it does not fire by spontaneous combustion, even when piled 85 ft. high. The largest installation of this type is at Abrams, Pa. Eight conical piles are made, each holding 60,000 short tons, on a base diameter of about 331 ft. and a height of 84⅓ ft. Astride of each conical pile is a steel shears, on one leg of which is a scraper conveyor which begins under a hopper below a railway track. Each leg of the shears is a latticed bow-string truss. The bottom of the trough of the inclined conveyor is a flexible steel band, which coils on a drum near the ground. Coal discharges over

the upper end of this band; by hauling it farther up on the shears the dumping point is advanced as the pile grows in height. This keeps the dumping point close to the growing pile, with a minimum fall.

For reloading, a radial, horizontal, lenticular arm made of structural steel is arranged between two adjacent conical piles, and mounted on wheels which move on concentric tracks. The sweep of this long arm covers the bases of the two piles. Carried in the arm is a reversible flight conveyor set on a horizontal axis so that the flights projecting outwardly attack the pile on tangents to the base circle. The arm is moved by power controlled at the pivot point. The reclaimed coal is carried in to the pivot point, chuted to an inclined scraper conveyor which elevates and carries the coal to a screening and loading tower for loading into coal trains. The reclaiming system will put out at each tower 180 tons per hour. On hillsides, a single traveling, bow-string, half-shears carries the inclined stocking conveyor. The head is mounted on a truck which runs on a single leg trestle with a continuous girder and rail. This will form a continuous pile as well as cones. With such machines the reloader moves longitudinally. This device has been applied to covered storage, to avoid the deterioration of coal exposed to sun and rain. They have been made to form conical piles under dome-shaped buildings of ingenious construction, and are also used for storing in rectangular buildings, one of which, for the Lehigh Valley Coal Co., stores 100,000 tons of egg, stove and nut coal under cover. This plant uses three stocking conveyors and reclaims by three tunnels containing Dodge open-top carriers. Each of the three units can dispose of 150 tons per hour. Another example of this interesting device may be seen at Shadyside, N. J., opposite the University, at the storage depot of the New York Edison Company.

Reciprocating conveyors have a limited use and require more power than flight conveyors, to which they are closely related. They have a stationary trough with swinging flights attached to a horizontal reciprocating frame, mounted on rollers. When the frame is thrust forward, the flights stand firm and push the material; on the return stroke, the flights fold up and slide out of and over the material. Acceleration is a factor at every stroke. They are used principally for sand.

GRADUATE COURSES IN HIGHWAY ENGINEERING AT COLUMBIA UNIVERSITY.

BY ARTHUR H. BLANCHARD.*

A technical graduate who is attracted to highway engineering has several more or less well defined fields open to him, namely: The highway departments of municipalities and towns; those of states, counties and parks; the engineering organizations of contractors; and the engineering and sales departments of companies dealing in materials and machinery used in highway work. In city and town work, matters relative to the construction and maintenance of streets and pavements compose the bulk of the work assigned to the highway departments, together with more or less road engineering problems. With state, county and park departments, the construction and maintenance of all types of road surfaces and bituminous pavements constitute 90 percent. of the work of such organizations, while certain problems in street pavements and highway bridges have to be dealt with occasionally. The prevailing idea, however, that the two fields just mentioned are easily separable and that the preparation for one should not be the preparation for the other is essentially wrong. Since the lines of demarcation between the above fields are rapidly becoming obliterated, the successful highway engineer of today, whether engaged by the city or state, must have a comprehensive knowledge of all branches of highway engineering and allied subjects. Otherwise it is obvious that it will be impossible to follow that important principle of economics of highway engineering, the adaptation of methods and materials to local conditions.

In contemplation of these opportunities, the essential prerequisites of a successful career as a highway engineer must be given due consideration in order that the future prospects offered by this field of engineering may be thoroughly understood.

The ideal foundation consists, first, of four years' training in a course in civil engineering; second, practical experience in both field and office in connection with the construction and maintenance

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of roads and pavements on a system of highways; and third, the acquisition of knowledge along certain lines of particular value to the highway engineer.

In explanation of the last prerequisite it might be stated that to be well informed, the highway engineer must acquire considerable knowledge relative to the economics of highway engineering, materials of highway engineering, management engineering, highway laws and systems of administration, mechanical appliances used in highway engineering, highway bridges and culverts. road and street surveying, drafting and designing, methods used in a road material laboratory, advanced dynamic and structural geology, lithology, petrology and petrography, processes of industrial chemistry, methods of testing bituminous materials and the interpretation of results, and finally advanced highway engineering covering the most recent practice throughout the world in the construction and maintenance of all kinds of roads and pavements.

The fulfillment of prerequisites numbers one and two is easily accomplished, except that the graduate of one or two years' standing may with difficulty retain his position during the months from December to March, inclusive, especially if he is connected with a state or county department or with the organization of a contractor. In these fields of highway engineering the immense amount of work to be completed during the construction season in the north requires the maximum engineering force obtainable both in the field and in the office, while during the four months mentioned above, the natural confinement of a large percentage of the work to the office necessitates reducing the engineering staff. The prospect of being without work for four months of the year has prevented many high-grade technical graduates from entering the field of highway engineering. In certain cases it has been possible, by coöperation between state highway departments and colleges giving courses in civil engineering, to mitigate the evils of this situation. As a concrete example, may be cited the writer's experience while he was Deputy Engineer of the State Board of Public Roads of Rhode Island. Many of the best civil engineering students at Brown University were employed throughout the college year part time and during vacations all the time, year in and year out, with the natural result that upon graduation some became

members of the permanent force. At all times, however, the office force consisted of trained men of a number commensurate with the work of a given season. It should be said that in many cases this plan will not work out satisfactorily for the field or inspection force, due primarily to the fact that while the construction season extends from April to November, inclusive, in certain sections of the country, the long vacations cover only June to September, inclusive, hence the impracticability of utilizing undergraduates, resident at the University, in the months of April, May, October and November in the above field positions.

The third mentioned prerequisite might, of course, be covered by collateral reading but it is self-evident that only a very limited idea of certain of the subjects mentioned can be acquired in this way. Especially is this statement applicable to over 50 percent. of the subjects which, it is apparent, must be illustrated and exemplified by laboratory equipment and well stocked museums, or developed through the medium of library research.

The problem before the educational institutions of this country is that of determining by what method the subjects outlined above can be offered upon a practical basis. The writer does not favor a four years' undergraduate course in highway engineering, not only because of the varied and potent reasons which have been advanced relative to over-specialized undergraduate courses, but also because undergraduate students are not sufficiently mature to acquire the benefits which should be derived from a combination of practical experience and specialized knowledge.

The most practicable plan is to arrange a definite course of instruction as a unified graduate course based on the assumption that the technical graduates enrolled for the master's degree shall hold undergraduate degrees in civil engineering. If the graduate instruction is given in the period from about December 1 to about April 1, it will be possible for practicing highway engineers, especially first- second- and third-year graduates, to use the winter period advantageously in acquiring advanced knowledge under favorable conditions.

It is gratifying to the highway engineering profession and that portion of the public interested in the development of good roads and streets throughout the United States that Columbia University

should have decided to establish graduate courses in highway engineering based upon a most comprehensive plan and upon the principles enunciated above.

The new graduate course offered at Columbia this year will cover the field of subjects referred to previously in this paper and will in amount be sufficient to satisfy the requirements for the master's degree. As the period of attendance will be from December to March inclusive, equivalent to about one semester, two periods of residence will be required to fulfill the requirements for the degree.

The program of the graduate courses offered at Columbia University is as follows:

FIRST YEAR.

Highway Engineering 101 — Economics and Design of Roads and Pavements.

Highway Engineering 103 — Bituminous Surfaces and Bituminous Pavements.

Highway Engineering 105 — Highway Bridges and Culverts.

Highway Engineering 107 — Mechanical Appliances used in Highway Engineering.

Highway Engineering 109 — Road Surveying and Design.

Highway Engineering 111 — Seminar in Highway Engineering Literature.

Chemistry H199 — Chemistry of Bituminous Materials.

Geology H215 — Engineering Geology.

SECOND YEAR.

Highway Engineering 102 — Macadam and other Roads; Block and Concrete Pavements.

Highway Engineering 104 — Highway Laws and Systems of Administration.

Highway Engineering 106 — Management Engineering.

Highway Engineering 108 — Road Material Laboratory.

Highway Engineering 110 — Street Surveying and Design.

Highway Engineering 112 — Seminar in Highway Engineering Literature.

Chemistry H200 — Chemistry of Bituminous Materials, Laboratory. Course.

Geology H216 — Lithology and Petrology.

Mineralogy H214 — Optical Mineralogy.

It is of interest to note that this plan has the enthusiastic support of many of the foremost highway officials and engineers in the United States. All emphasize the feasibility of granting four months' leave of absence to practically all the young civil engineers who wish to take graduate courses in highway engineering. It is the hope of those interested in the higher education of highway engineers in the United States that it will be possible in the near future to lay the foundation for the establishment of a corps of highway engineers comparable to that admirable body of trained men who have graduated from l'Ecole Nationale des Ponts et Chaussées of France.

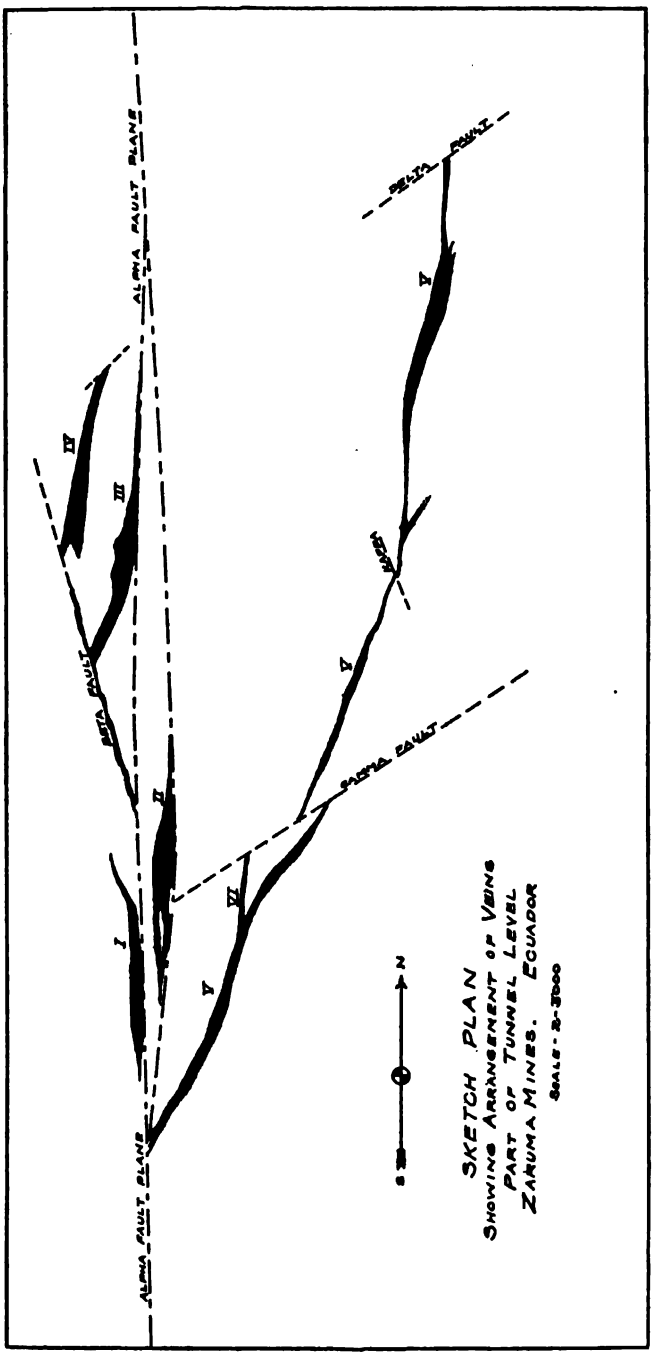
NOTES ON THE GEOLOGY OF THE ZARUMA MINES, ECUADOR.

BY J. G. BARAGWANATH.

The gold mines of Zaruma are situated near the town of Zaruma, Province of El Oro, in southwest Equador, 3° 45′ south of the equator, and two days' journey from tide-water of the Pacific, and east of the cordillera of Dumari, which forms the western rampart of the Andes. The topography of the section is of the deeply dissected type so common in tropical regions of heavy rainfall. The great valley lying between the two cordilleras has been eroded into a succession of deep *quebradas* with steep, precipitous slopes, leaving sharp, hog-back ridges between which make travel exceedingly slow and difficult.

North of the mines are great masses of rhyolite and andesite porphyries, and diorites with some granites, which cover a wide area. These comparatively recent igneous rocks have been thrust up through, and spread out over, older mica-schists which appear to the south. These schists are of uncertain age but appear to be of sedimentary origin, though their original characteristics have been so altered by the dynamic strains involved in the elevation of the Andean ranges that such a statement cannot be definite. The eruptives overlying these schists present a complex and infinite variety of felsitic and rhyolitic rocks in flows, sheets and laccoliths. They have been extensively faulted and are intersected by numerous veins of quartz, calcite, and barite, most of which are more or less auriferous. They weather rapidly, and, in the less exposed portions, decomposition has extended to great depth. This decomposition results in a sticky clay, ranging in color from yellow to deep red. Certain of the porphyries rich in feldspars, especially near Zaruma, upon decomposition yield a fine white clay which seems to be of excellent grade.

Old workings, dating, many of them, from Spanish times are common all through the region. Everywhere one finds traces of old tunnels and shafts as well as open-cuts of surprising size. Pres-


ALPHA FAULT PLANE
BETA FAULT
GAMMA FAULT
DELTA FAULT
I
II
III
IV
V
VI
VII
N
SKETCH PLAN
SHOWING ARRANGEMENT OF VEINS
PART OF TUNNEL LEVEL
ZARUMA MINES. ECUADOR
SCALE: 2-3000

ent operations are being carried on at Mina Portovelo and Mina Grande, two connected mines, by American capital.

The veins being worked are in a mass of andesite in which the phenocrysts of plagioclase are strongly enough developed to make it, in certain portions, an andesite porphyry. Near the veins the feldspar is usually partially replaced by iron pyrites and sometimes quartz. In places this replacement by quartz has been so extensive that other minerals beside the feldspars have yielded to it, and we find many masses of this indurated country rock which have lost entirely their original characteristics and are now white and flinty and make very hard drilling. A few pyrite crystals can always be seen scattered through the country rock, but in the vicinity of a vein these become thicker, until quite close to the vein we have a heavily mineralized country rock usually carrying gold. Curiously enough, while in these veins we find the sulphides of iron, zinc, lead and copper, only the iron is found disseminated in the wall rock.

The accompanying sketch will convey some idea of the occurrence of the orebodies in which operations are now being conducted. The veins are associated with a strong and extensive fault which I have called the alpha fault plane. Neither the direction nor the extent of the displacement which accompanied this great fault movement is definitely known. A heavy gouge is always associated with the fault as well as small stringers of calcite and sometimes quartz, though very little brecciation is evident. This fault has been the chief factor in the formation of the various orebodies of the alpha system which lie along it. In many places it has split into series of parallel faults which again converge into one main fault. These splits offered an excellent opportunity for vein formation of the true fissure type, and we find them wherever this phenomenon occurs, the small stringers of calcite and quartz widening out into large veins. Veins I, II, and III are of this type, and many more appear to the south. Where two veins lie along the fault plane, opposite each other, the one nearest the fault is mostly calcite while the other is quartz, though this seems to have no influence on their value.

The alpha fault plane, with its associated orebodies, has a normal dip of 71° E., except between its junctions with vein V and beta fault, where it is almost vertical. Everywhere countless smaller

slips radiate from the main fault in all directions and at all dips, but are of little importance except to make the geology more confused. The gouges never carry appreciable values.

Beta fault, which is also a quartz vein, abuts against the alpha fault, but does not cross it. It clearly displaces veins III and IV. As almost all of the faults thus far have proved to be thrust faults, we may safely infer that beta is of this kind and the other portions of veins III and IV will eventually be picked up by a crosscut to the west. This is a good illustration of an older vein faulted by a later one.

Quite distinct in many ways from the veins of the alpha system is vein V. This is a true fissure vein exhibiting quite perfect banding and a large variety of minerals. The most common of these are quartz, sphalerite, pyrite, and galena, but rhodonite, chalcopyrite, malachite, azurite, calcite, bornite, tetrahedrite, and a little native copper are occasionally seen. From the alpha fault plane north to kappa, a small reversed fault, the vein is nearly vertical, but from kappa north it has a dip of 68° E., and a slight change of strike. The vein is conspicuously faulted by gamma, an important slip which, like beta, does not cross the fault plane of alpha. Gamma is a cleancut, almost vertical fault. Vein V is again faulted by delta and has not been picked up to the north, though it would be logical to suppose that a crosscut to the west would discover the continuation of this valuable vein.

Vein VI is a branch of vein V, a vertical deposit of the fissure type with a north and south strike, lying along a hanging-wall slip. Though clearly a branch of vein V it has none of the mineralogical characteristics of the former, being filled with calcite and quartz. It is generally high in gold. Not far from its junction with vein V it is cut off by gamma fault and pinches out beyond into small stringers of no value on this level. In no instance do the values continue for more than a few meters north of the fault, so that for all practical purposes gamma fault is the northern limit of the vein. Below this level the veins V and VI incline toward each other slightly. Their junction makes downward to the north so that vein VI becomes shorter as we go down and we would expect it to pinch out in depth.

Vein IV, lying west of and roughly parallel to vein III, is faulted

by beta which forms its only known southern extremity, and is cut off by a small slip to the north. Curiously, the vein exhibits the same mineralogical character as No. V, and has approximately the same strike and dip. Furthermore, the veins of the alpha system are distinctly quartz and calcite veins with a dip of 71° E., and invariably lie along the alpha fault, while No. IV bears no relation to this fault. Although vein IV has been considered a separate deposit these facts seem to point to its having once been the southern continuation of vein V, which has been thrown into its present position by the enormous *thrust* movement of alpha. Unfortunately no exploration work has been done to discover whether the faulted northern part of the vein joins alpha fault plane, a condition which would strongly support this theory.

Supposing this to have been the case, then veins IV and V are one and the same, an older vein than alpha fault and its associated veins. No. VI is clearly of the same age as No. V, as its vein matter blends without break into that of No. V. Gamma, delta, and kappa, while more recent than vein V, may or may not be later than alpha. Beta vein, however, cutting vein III, a member of the alpha system, is the youngest of all.

Veins V and VI are generally higher grade than the alpha series. In all veins the gold usually occurs in distinct shoots pitching sharply to the north. There is nothing in the vein structure to indicate the location of these shoots, and their limits can only be determined by assay. Gold is found free, disseminated through the quartz and calcite, and also locked up, probably mechanically, in the sulphides. So fine is this dissemination that $10 ore will often not yield a color in the pan, and much richer rock only a few, very small particles. Manganese is present in all the ore, though the form in which it occurs is not known. It is as widely and minutely distributed as the gold and may have acted as a precipitating agent for the precious metal. Free gold is rarely visible to the eye, but when it is, it appears as an aggregation of small particles, always in conjunction with some dark mineral, probably a silver sulphide and some manganese mineral. No silver minerals can be recognized, but enough silver is usually present to permit a bead to be parted, and the final bullion contains about 30 percent. of the metal.

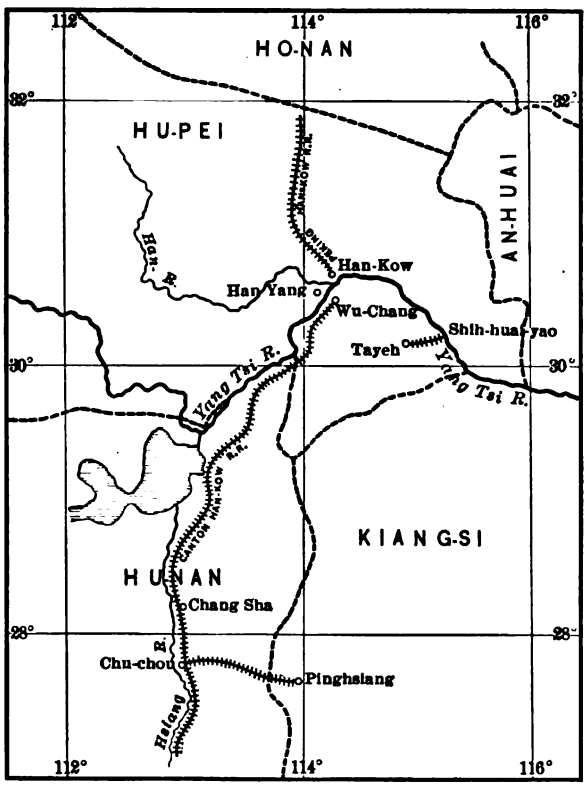
NOTES ON THE HAN-YEH-PING IRON AND COAL COMPANY, CHINA.

BY CHO-YANG AND CHENG-FU WANG.

This company's operations are conducted at three chief plants:

1. Han-yang iron and steel works.
2. Ta-yeh iron mines and limestone quarry.
3. Ping-hsiang coal mine and coking plant.

The location of these plants is shown on the accompanying sketch map (Fig. 1).



LOCATION OF HAN-YEH-PING COMPANY'S OPERATIONS.

The Han-yang iron works were established in 1891 by the late Grand Counsellor Chang Chih-Tung, the iron mining company in

1897, and the coal mining and coking plant in 1898. Since then, the company has recorded progress every year. Besides these three plants mentioned, this company owns also some manganese and small iron-ore mines in Hunan province.

HAN-YANG IRON AND STEEL WORKS.

The Han-yang iron and steel works is situated on the northern bank of the Yang-tse river, being separated from Hankow by the Han river, and from Wu-chang, the capital of Hupei province, by the Yang-tse river.

The iron works consists of three modern blast furnaces and produces about 500 tons of pig iron every day, or 180,000 tons per year. The height of the two old furnaces is about 70 ft. and the external diameter 20 ft. About two-thirds of the pig iron produced goes to the steel plant for making rails, angles, beams, plates, sheets, and all sorts of railroad supplies. A small portion is used in the foundry and some is sold to the native works of the surrounding cities. The rest of the pig iron is exported to Japan and United States; it is said that the price of Han-yang pig iron in New York is less than that of Pittsburg. Table I states the composition of the pig iron and of other products made by this company.

TABLE I. ANALYSES OF HAN-YANG PRODUCTS.*

	Foundry Pig-iron	Basic Pig-iron	Open-hearth Iron	Finished Steel
C.	3.8	3.9	3.34	0.40
Si.	2.5	0.65	—	0.085
Mn.	1.0	1.10	0.26	0.089
S.	0.01	0.045	0	0.055
P.	0.02	0.25	.45	0.028

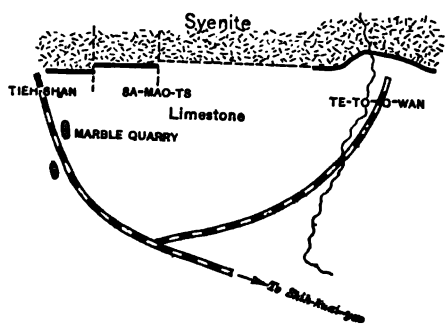
The steel plant consists of six Siemens-Martin open-hearth furnaces, one mixing furnace, a cogging mill, beam and angle mill, rail mill, plate mill and some other plants capable of rolling 1000 tons of finished product per day. At present the output of steel is about 300 tons per day.

* *Eng. and Min. Jour.*, June 11, 1910, p. 1233.

Ta-yeh Iron Mines.

Ta-yeh is situated about 90 miles down the Yang-tse river from Hankow, and 30 miles southward.

The ore occurs as a contact deposit between dark syenite and marble (See Fig. 2). The main orebodies appear in three hills of about 800 ft. height. Two of these hills are quite near to one another, being separated by a narrow valley, while the third is at a distance of 3.5 miles east. The syenite, which is very hard and compact and not much affected by weathering, lies against the orebody. The marble, however, has been greatly eroded and in many places



Occurrence of Iron Ore at Ta-yeh.

the ore is left exposed to the surface. The vein runs about N. 75° W., and is nearly vertical, dipping slightly to the north. Its thickness is from 100 to more than 200 ft.

The ores consist mainly of hard hematite, some magnetite, in Sa-moa-ts, and limonite to small extent. The formation of magnetite may be due to reduction by copper solution. Near this place we found a cave quite rich in copper sulphate and carbonate. Most of the mining is done in the hematite ore by open cut. The ore is blasted down as in quarrying stone, the drilling being all done by hand. At present, very little stripping is performed. After the ore is blasted down, it is sent down by gravity planes in one-ton cars, and dumped by means of revolving cradles into 11-ton railroad cars. Each train carries a dozen of these cars to the river side, which is known as Shih-huai-yao, and there the ore is loaded into lighters, each of which has a capacity of 350 tons. The lighters are towed to Han-yang by a steamer which takes two of them each trip.

VIEW OF PING-HSIANG.

HEADFRAME, PING-HSIANG COLLIERY.

COKE OVENS, PING-HSIANG COLLIERY.

TABLE II. APPROXIMATE COST OF PRODUCING IRON ORE IN THE OPEN-CUT WORKINGS AT TA-YEH.

Stripping	Mexican	pesos	0.08	per	ton
Mining	"	"	0.18	"	"
Tramming	"	"	0.03	"	"
Powder, steel, etc.	"	"	0.015	"	"
Superintendence	"	"	0.06	"	"
Loading cars, freight to Yang-tse, etc.	"	"	0.30	"	"
Total			0.665		

The output at Ta-yeh is 1600 tons daily, or about 500,000 tons per year. Of this, 125,000 tons is exported to Japan every year. At the present rate of output, the ore in sight will last some 70 years to come. The amount at greater depths is not known.

Table III presents analyses of the three principal types of ore from these mines.

TABLE III. ANALYSES OF TA-YEH ORES.

	Hematite.	Brown Ore.	Magnetite.
Fe	62.0	48.0	65.0
Mn	0.3	7.0	0.13
SiO_2	4.0	6.0	3.0
Al_2O_3	2.0	—	—
S.	0.02	0.01	0.06
P.	0.05	0.03	0.12

Ping-Hsiang Coal Mine and Coking Plant.

Ping-Hsiang is located near the mid-west boundary of Kiang-si province, and is about 250 miles south of Han-yang. The coal mine is four miles southeast of Ping-Hsiang city.

The country rock of this mountainous vicinity is red sandstone. The coal outcrops on the hillside. There are three divisions, the upper, the middle and the lower, containing seams running from 6 in. to 36 ft. thick. In quality the coal ranges from soft to hard bituminous.

There were three shafts and a drift in 1908. The drift runs about two miles eastward and along the strike, at a general dip of 10°. This main drift is arched with bricks, and transportation is by electric locomotives. One shaft is 560 ft. deep, and the ventilation shaft is 300 ft. Their shape is elliptical, about 15x13 ft., with brick lining.

Mining is carried on by the longwall system, and the coal is taken out when advancing. The roof is allowed to fall, and the haulage roads are framed for excessive pressure by means of double roof timbering. Linings are either of timber or of brick arches. Drifts for ventilation are opened every 20 to 25 meters along the haulage way, and connected at about 20 meters distance from the main gangway.

Analyses of the coal and coke are presented in Table IV.

TABLE IV. ANALYSIS OF PING-HSIANG COAL AND COKE.

	Coal		Coke
Ash	10 — 25 %	Ash	10 — 25%
Vol. matter	25 — 35 %	S.	0.3 — 0.6%
Sulphur	02 — 0.6%	P.	.06 — 25%

The present output is 600,000 tons per year, which is expected to increase to 1,000,000 tons. The present assumption for this field is said to be 300 million tons, which will last some 200 years to come. Coal brought from the mine is screened on a 50x80-m.m. screen and the lump coal is either crushed in rolls and re-screened, or sent directly to railroad cars for market. The final cleaned coal from the washing plant is used for making coke.

TABLE V. WAGES OF LABORERS.

Iron and Coal Mines.			
Miners	Mexican pesos	0.25— .30	per day
Loaders and pushers	" "	0.10— .15	" "
Tipple coolies	" "	0.10—.125	" "
At Steel Works.			
Rollers on Mills	Mexican pesos	0.35— .70	per day
Heaters on furnace and open-hearth smelters	" "	0.55	" "
Helpers	" "	0.35— .50	" "
Blacksmith	" "	0.70—1.00	" "

There are two sest of ovens in action, comprising 300 retorts. They are of Otto-Koepe type, which is very similar to the Otto-Hoffman. The by-product is used to heat the retorts and the boilers. Coking lasts 72 hours. In 1908 the production was 180,000 tons. This was mainly transported to Han-yang for the blast-furnace supply, first by railway to Chu-Chow and then by lighters to Han-yang.

ABSTRACTS—ANALYTICAL CHEMISTRY.

BY E. WALLER.

APPARATUS—REAGENTS—MANIPULATION.

Bumping of Solutions.—Spurrier. (*J. Am. Chem. Soc.*, XXXIII, 1632). In boiling solutions covered with a layer of oil, bumping will often occur, but may be prevented by dropping in a piece of tubing open at both ends, but long enough for the upper end to rest against the side of the vessel above the level of the liquid. As spurts of the liquid constantly pass up the tube, the free end should not extend so high that portions of the solution might be projected from the vessel, and be lost.

Zinc Oxide in Glassware.—Smoot. (*Eng. and Min. J.*, XCI, 1098) Some forms of refractory glass contain ZnO. Such glass resists the acid of water and acid solutions of salt, but gives up Zn to fixed alkalis or alkaline solutions of salts; ammoniacal solutions have scarcely any effect.

Ether in Metal Analysis.—Mylius and Hüttner. (*Ber.*, XLVI, 1315) The use of ethyl ether for extraction and separation of metallic chlorides has been studied, and a table prepared showing the proportions of chlorides extracted by shaking out with ether. The proportion of free HCl has a marked effect. With most metals the maximum extraction with the ether is produced when 20 to 22 percent. of free HCl is present. With $SbCl_3$ the maximum extraction is at or about 10 per cent. free HCl; with $HgCl_2$ the free HCl exercises a preventive effect, more $HgCl_2$ being extracted from the neutral aqueous solution, than when acidified. The phenomena are more or less modified by the presence of alkaline and other chlorides which are capable of forming double salts. Practically, Fe and Au only can be separated from certain other metals as chlorides by this means.

Ether—Test for Water.—Tyrer. (*Proc. Chem. Soc.*, XXVII, 142) A determination of the degree of solubility of CdI_2 in the sample will indicate the proportion of water present. The solubility is given as follows:

Gm. H_2O in 100 of ether.	Gm. CdI_2 per 100 of solvent.	Gm. H_2O in 100 of ether.	Gm. CdI_2 per 100 of solvent.
0.	0.143	0.90	6.46
0.10	0.78	1.00	7.30
0.30	2.07	1.10	8.27
0.50	3.36	1.14 (Sat.)	8.68
0.70	4.77		

Eder has stated that CdI_2 is soluble in "absolute" ether to the extent of 28 per cent. (*Dingl. Polyt.*, CCXXI, 189). He must have used a wet ether.

Sulfur Monochloride for Decomposing Rare Earth Minerals.—Hicks.

(*J. Am. Chem. Soc.*, XXXIII, 1492) Experiments were made with fergusonite, eschynite, euxenite and samarskite. The pulverized minerals were placed in a porcelain boat in a combustion tube, which was heated in a furnace while the monochloride vapors were passed over them. The volatile chlorides and excess of monochloride were passed into dilute HNO_3. From this solution the excess of S was removed by adding excess of NH_4OH and H_2S. The method is not claimed to be new, but is strongly recommended on account of the ease of decomposing these minerals, the simplicity and cheapness of the apparatus, and the ease and completeness with which Cb, Ta, Ti and W are separated from the rare earths by the operation.

Ammonium Citrate for Phosphate Analyses.—Hull and Bell. (*J. Am. Chem. Soc.*, XXXIII, 711). The preparation of an "exactly neutral" solution of citrate is not easy and has frequently been the source of disagreements in results. The authors find that when "exactly neutral," the electrical conductivity, as also the specific gravity, are at a maximum. Also that free NH_3 can be detected by shaking with chloroform, in which neither citric acid nor ammonium citrate is soluble.

Sodium Paratungstate in Determinations by Loss.—Gooch and Kuzirian. (*Am. J. Sci.* XXI, 497) The reagent is prepared by fusing WO_3 with an equal weight of fused Na_2WO_3. 2.5 gm. of this, when fused with 0.5 gm. of an alkaline or alkaline earth carbonate or nitrate, will quantitatively expel CO_2 or N_2O_5 from its combination.

Standardizing Alkalimetric Solutions.—Feld. (*Zts. angew. Chem.*, XXIV, 1161) $Na_2S_2O_3.5H_2O$ can now be obtained in a very pure state, to be weighed out for this purpose. With an *excess* of $HgCl_2$ (saturated solution) the reaction is: $Na_2S_2O_3+2HgCl_2+H_2O=Hg_2Cl_2+Na_2SO_4+2HCl+S$. *After* the reaction is complete, an excess of $4N/NH_4Cl$ solution is added (to prevent separation of HgO with the alkali) and the solution is titrated with the alkali to be standardized, using methyl orange. If the $HgCl_2$ is not in excess, HgS will separate, and mask the color of the indicator.

Small Amounts of Lime in Presence of much Magnesia.—Murrmann. (*Monatsh.*, XXXII, 105) If the ratio MgO:CaO is less than 50:1 the best procedure is: Neutralize, add 3 c.c. strong NH_4Cl solution, and 3 drops of HCl, and heat to boiling. Then add concentrated solution of $H_2C_2O_4$ and 3 or 4cc. of quinoline. After 10 minutes standing, the CaC_2O_4 may be filtered off. Pyridine may be used in place of quinoline, but a more exact neutralization (with aid of methyl orange) is necessary. Use a 2-percent. solution of pyridine, and add slowly, stopping the addition before the indicator shows the positive yellow color. The pyridine can be boiled out of the filtrate, and the MgO determined therein.

Qualitative for Chloride in Presence of Bromide or Iodide.—Testa. (*L'Ind Chim.*, XI, 123) AgCl is slightly soluble in boiling solution of $(NH_4)_2C_2O_4$. AgBr is much less soluble and AgI not at all. Therefore on boiling the washed precipitate of Ag salts for a minute in N/2 oxalate solution, filtering, and adding KBr to the clear filtrate, the presence of chloride is indicated by the occurrence of turbidity.

ORES—METALLURGICAL (NON-FERROUS) PRODUCTS.

Commercial Aluminum.—Kleist. (*Chem. Ztg.*, XXXV, 668) In one portion, dissolved in a mixture of HCl, HNO_3, and H_2SO_4 and evaporated to SO_3 fumes, the Cu and Sn are precipitated together by H_2S (from a clear solution), ignited and weighed together as oxides. Another portion (1 to 2 gm.) is dissolved by boiling with 150 c.c. HCl in an Erlenmeyer, and the hot solution is titrated with a standardized solution of $FeCl_2$. The end point is a lemon-yellow turbidity in the solution. This determines Sn. The Cu may be estimated by difference or electrolytically. For Zn, after removing Sn and Cu by H_2S, concentrate to 100 c.c., render ammoniacal, and then add H_2SO_4 until the $Al_2(OH)_6$ is just redissolved; then dilute to 400-500 c.c. and precipitate Zn by H_2S. Redissolve in dilute HCl and reprecipitate in the same manner before separating the Zn for weighing.

Iron and Alumina Separation.—Kharich Kov. (*Chem. Ztg.*, XXXV, 463) Ferrous iron is completely extracted from its solution, when slightly acid or neutral, by agitation with solution of naphthenic acid in benzol or in petroleum ether. The organic solvent assumes a chocolate brown color on dissolving the Fe, which must be in the ferrous condition. The application of the method for separation from Al is said to be satisfactory.

Iron Titration.—Koste. (*Zts. angew. Chem.*, XXIII, 2354) In the analysis of German silver, after separating Fe by NH_4OH and redissolving in HCl, a few drops of 5-percent. $CuCl_2$ are added, then 15 to 20 c.c. of a 10-percent. solution of sodium salicylate, and the Fe is titrated with standard $Na_2S_2O_3$ until the color is discharged. Great exactness is claimed for this method. The $Na_2S_2O_3$ is standardized by a 1-percent. solution of Fe_2Cl_6.

Antimony in Alloys. Volumetric, Rapid Method.—Jamieson. (*J. Ind. Eng. Chem.*, III, 250) The reaction is: $2SbCl_3+KIO_3+6HCl=2SbCl_5+KCl+ICl+3H_2O$. The process requires a solution of KIO_3 containing 3.5667 gm. per litre (1 c.c.=0.004 Sb.) and of ICl, which is made by dissolving 10 gm. KI and 6.44 gm. KIO_3 in 75 c.c. H_2O, and adding 75 c.c. conc. HCl. A little chloroform is then introduced, and the solution adjusted to a faint color of I in the $CHCl_3$ by adding KI or KIO_3 as may be found necessary. Frequent shaking is of course necessary in this process. The amount of alloy to be taken will vary with the proportions of Sb. If under 2 percent., 1 gm. or more is necessary. If very high, 0.1 or 0.2 gm. may suffice. Ordinarily about 0.5 gm. is a good amount. Dissolve in 10 c.c. conc. H_2SO_4 and boil gently until decomposed, and the $PbSO_4$ shows white. Dilute with 15 c.c. H_2O, cool, and add 15 c.c. of 1:1 HCl. Filter through asbestos in a Gooch, and wash with the 1:1 acid, running filtrate and washings into a stoppered flask of about 250-c.c. capacity. Now add 5 c.c. of chloroform, and 15 c.c. of conc. HCl (to compensate for dilution by the titrating solution.) Let stand about 5 min., and then titrate until the $CHCl_3$ is just decolorized, after thorough shaking and standing for about a minute. If much over 15 c.c. of the standard solution is used, some conc. HCl should be added to keep the strength near the 1:1 point, which is

an indispensable condition. As is rarely present in such alloys, but if present, it reacts in the same manner as the Sb. It can be removed by H_2S in the HCl dilution of the solution obtained as above, passing air through afterward, and then filtering off, before titrating.

Tin and Antimony in White Metals.—Beneker. (*J. Ind. Eng. Chem.*, III, 637) Dissolve 0.5 gm. in a mixture of 25 c.c. HCl and 5 c.c. HNO_3, or in 25 c.c. HCl saturated with Br (not Br water). Warm, and add 1 gm. tartaric. Dilute with hot water to 400 c.c., and slowly add excess of NaOH. Then add about 20 c.c. of a 10-percent. solution of NaHS. Make this reagent by saturating 10-percent. NaOH with H_2S, and then adding an equal bulk of 10-percent. NaOH solution. Wash twice by decantation, then on the filter, using a diluted Na_2S solution for washing. The residue is dissolved in HNO_3 and treated for Pb, Cu, etc. Acidify the filtrate slightly with HCl, filter and wash off all H_2S. Put the filter containing the sulfides in an Erlenmeyer to which is fitted a cork carrying a swan neck thistle tube. Add a measured excess of N/10 I solution, 30 to 50 c.c. of conc. HCl, and about 2 gm. tartaric. Stopper and boil for a few minutes.

The Sb dissolves as SbI_3, the Sn as SnI_4. Cool, dilute, and titrate the excess of I with standard $Na_2S_2O_3$. Rinse the solution into a beaker, neutralize with Na_2CO_3, add an excess of $NaHCO_3$, and titrate Sb with N/10 I. Multiply the c.c. of N/10 I used in this case by 3/2, and deduct the product from the c.c. of N/10 I consumed in the first titration. The remainder is the number of c.c. to be calculated to Sn. $SbI_3+I_2=SbI_5$. $SnS_2+I_4=SnI_4+2S$. The free S does not interfere.

Arsenic and antimony in commercial copper.—Kern and Wen. (*Met. and Chem. Eng.*, IX, 365) Dissolve 10 to 30 gm. in HNO_3, then add about 1 gm. $FeSO_4$, and after boiling dilute the solution and precipitate by NH_4OH. The precipitate is dissolved in HNO_3 and H_2SO_4, evaporated to fumes, and hypophosphite added. After heating to strong fumes for some time the solution is cooled and transferred to the distilling flask, using very little water for rinsing. Add 40 c.c. of HCl. The flask is fitted with a stopcock bulb containing 80 c.c. of HCl, which is allowed to drop in slowly as the distillation progresses. The distillate is neutralized with KOH and $NaHCO_3$, and titrated with N/10 I. From the liquid remaining in the flask, the Sb is precipitated by H_2S, dissolved in HCl and $KClO_3$, and eventually treated with KI and the free I titrated with N/10$Na_2S_2O_3$, calculating to Sb.

Tin in Ores.—Lewis. (*Lond. Min. J.*, June 10, 1911, 606.) Weigh out an amount of the pulverized ore according to probable richness; extract by digestion with HNO_3; dry; and reduce to metal by ignition in a current of H or of coal gas. Dissolve out the metal by HCl, oxidize with a few drops of HNO_3, filter, and add NH_4OH until just alkaline. Then redissolve the precipitate by adding a hot solution of 3 gm. $NH_4HC_2O_4$ with 7 gm. $H_2C_2O_4$. The solution, which should now be about 100 c.c., is heated to boiling, and H_2S passed in for an hour. Filter and wash with as little H_2S water as possible. The solution now contains Sn free from contaminants, with the possible exception of Fe. Boil for some time to remove H_2S and electrolyze. For this stage of

the treatment, not over 0.4 gm. Sn should be present. The Sn obtained should be dissolved in HCl and tested for Fe. If any is present, it should be determined and deducted.

Gold in Alloys.—Mylius. (*Zts. anorg. Chem.*, LXX, 203.) For the separation of Au in analyses of coins and bullion, the ether separation of $AuCl_3$ from a solution containing 5 to 10 percent. free HCl has been found to be most effective and accurate. *Per contra*, it also affords a means for obtaining Au of a high degree of purity.

Parting Platinum-Gold-Silver Bullion.—(*Eng. Min. J.*, XCII, 259.) Silver must be added until the alloy is 75 percent. Ag. Roll into a thin sheet. Dissolve Ag by boiling with conc. H_2SO_4 in the proportion of 2 to 3 oz. per ounce of alloy. Pour into water and wash residual Au and Pt. Disolve those in aqua regia; evaporate down; separate Pt by use of NH_4Cl, and in the filtrate obtain Au by $FeSO_4$.

Silver—Colorimetric.—Whitby. (*Zts. anorg. Chem.*, LXVII, 62.) The author utilizes the brown coloration given by Ag ions with fixed alkali solutions containing organic substances, such as dextrin, starch, sugar, etc. The mode of application consists in adding a few drops of conc. solution of sugar to 50 c.c. of the Ag solution. Heat on the water bath for 2 minutes, then add about 6 drops N/NaOH and heat again not over 2 minutes. Cool and compare in Nessler tubes.

Lead by the Chromate Method.—Waddell. (*J. Ind. Eng. Chem.*, III, 638) This is a recommendation of the method described by Guess. (*Trans., A. I. M. E.*, XXXV, 359, 1905) It consists in decomposing with HNO_3 and HCl, evaporating the major part of the free acid, neutralizing with NH_4OH, adding acetic and $NH_4C_2H_3O_2$, and precipitating with $K_2Cr_2O_7$. From the mixed insoluble and $PbCrO_4$ the latter is dissolved out by 1:1 HCl, and the CrO_3 determined by adding KI and titrating with standard $Na_2S_2O_3$, using starch indicator. Investigation showed that washing the mixed precipitate and residue with 0.5-percent. acetic dissolved some $PbCrO_4$, which was prevented by the use of a little $K_2Cr_2O_7$ in the washing solution, one final wash with clear water being given. Preliminary to titration the solution (200 c.c.) should contain over 25 c.c. of 1:1 HCl (preferably about 50 c.c.). One gm. of KI is the best amount to add. For materials low in Pb the process is claimed especially to be superior to the molybdate method.

Lead in Ores.—Merrill. (*Eng. and Min. J.*, XCI, 569) The author claims that the ordinary method seldom, if ever, determines all the Pb. After decomposing with HCl and HNO_3, evaporating to fumes with H_2SO_4, diluting and filtering, the filtrate contains some Pb which should be precipitated out with Zn, converted to $PbSO_4$ and added to the main lot. In titrating with molybdate, the use of a *freshly prepared* solution of tannic acid (1 percent.) is recommended.

Lead—Higher Oxides.—Chivala and Colle. (*Zts. Anal. Chem.*, L, 209) The authors find more or less error in all the methods used. With Pb_3O_4 the errors are in some cases small, but are much greater with PbO_2. Iodometric methods are often very inaccurate, due to the formation of HIO_3, or in case acetates are present to formation of

iodacetic, or to partial oxidation of the acetic to CO_2. Lux's oxalic method seemed much better, though no method is absolutely recommended. As modified by the authors, the method runs thus: To 1 or 2 gm. of the sample add 100 c.c. N/HNO_3 and 25 c.c. N/oxalic; boil until completely decomposed (10 to 15 minutes). Titrate at 70 to 80° C. with standard $KMnO_4$, running the major part in rapidly, then let stand 5 minutes and finish by adding it drop by drop.

Copper in Pyrites and Slag.—Ivanov. (*Chem. Ztg.*, XXXV, 531) If the material is mixed with an equal weight of finely divided Fe, and heated to dull red for a few minutes, the Cu is obtained in a form readily soluble in HCl, from which solution it can be precipitated (by H_2S or other reagents) and dissolved for electrolytic estimation.

Copper—Methods of the Anaconda Copper Mining Co.—Traphagen. (*Western Chemist and Metallurgist*) The Cu is invariably determined electrolytically (about 0.11 amp. and 1.4 to 2.3 volts). In the presence of Bi, As, Sb, Se or Te, the Cu is separated by CNS method, or by adding 0.100 gm. Fe, barely neutralizing with NH_4OH, and boiling to precipitate, the precipitation being repeated two or three times. In mattes or converter copper, the Ag determined by fire assay is deducted from the Cu+Ag deposited on the platinum cylinder. Or the Ag may be separated before electrolyzing by adding just enough NaCl solution to precipitate it, and filtering.

Converter Copper.—0.5 gm.; 8 c.c. HNO_3; 8 c.c. H_2O; 1 c.c. H_2SO_4. Evaporate to fumes, dilute and electrolyze.

Mattes.—Wet 1 gm. with H_2O, add 8 c.c. HNO_3 and 1 c.c. H_2SO_4. Evaporate to dryness, take up with 8 c.c. HNO_3 and H_2O. Filter and electrolyze.

Slags.—Treat 2 gm. in platinum dish with 8 c.c. HNO_3, 8 c.c. HF and 1 to 2 c.c. H_2SO_4. Evaporate to fumes. Take up with H_2O and 10 c.c. HNO_3.

Sulfide Ores.—Treat 1 gm. with 8 c.c. HNO_3 and a little $KClO_3$. Evaporate to complete dryness, take up with H_2O and 6 to 10 c.c. HNO_3.

Oxidized Ores.—Treat 1 gm. with 8 c.c. HNO_3. Evaporate to dryness. Then add 10 c.c. HCl and 2 c.c. H_2SO_4 and evaporate to SO_3 fumes. Take up with 8 c.c. HNO_3 and water.

Cadmium in Spelter.—Fairlie. (*Metal. Ind.*, VIII, 386 and 511) Treat 20 gm. of the sample with 100 c.c. H_2O and only enough HCl to dissolve all but about 1 gm. of the metal; the residue contains all the Cd, and some of the Pb. Filter off, wash, dissolve in HNO_3, precipitate out Pb by H_2SO_4, neutralize the filtrate with NH_4OH, acidify slightly with HCl, and precipitate the Cd with H_2S. The CdS may be dried at 100° and weighed as such, or may be warmed with $Fe_2(SO_4)_3$ solution and then titrated with permanganate.

Bismuth in Ores.—Frerichs. (*J. Ind. Eng. Chem.*, III, 774) The ore should be ground to 100 mesh. Treat 0.5 gm. in a flask with 10 c.c. conc. HNO_3; boil until the acid is nearly gone, add 10 c.c. conc. HCl, and heat until it is practically decomposed. Dilute with 10 c.c. of water, filter, wash with dilute HCl (1:3), nearly neutralize with NH_4OH, dilute to 300 c.c., and pass H_2S. Filter and wash with hot

water. [If As or Sb are present, separate them at this point by treatment with $(NH_4)_2S$] Boil precipitate, filter and all, with 10 to 15 c.c. HNO_3 (1:1) until the filter is disintegrated, and the separated S is white. Dilute, filter into a 250-c.c. flask, washing with dilute HNO_3 (1:1). At this point the treatment differs according to whether the electrolytic or the basic chloride method is to be used.

Electrolytic.—Boil down to fumes with 6 c.c. conc. H_2SO_4, cool, add 50 c.c. water, cool, filter off $PbSO_4$, washing with a mixture of 6 c.c. conc. H_2SO_4 and 75 c.c. water. Dilute filtrate and washings to 200 c.c. and electrolyze at 0.5 to 0.6 amp., increasing to 1 amp. at the end. For electrolysis, the solution should not contain over 0.03 gm. Bi; with larger amounts, the metal may not adhere to the cathode. The anode must also be weighed before and after, as some small amount of Bi_2O_5 may attach itself thereto. The weighing should be made on a balance sensitive to 0.00001 gram.

Basic Chloride.—Dilute to about 250 c.c. Heat nearly to boiling and add 10-percent. NH_4OH to slight opalescence. Dilute to 350 c.c., add 1 c.c. HCl (1:3) and 2 to 3 c.c. saturated NH_4Cl. Cover and keep *nearly* at boiling until the BiOCl separates well. Filter and wash with hot water. The first precipitate may contain a little Pb. Dissolve in HNO_3 (1:1) and reprecipitate in the manner just described. Filter on a weighed filter, dry at 110° C., and weigh.

Nickel—Separation from Zinc in German Silver, etc.—Spring. (*J. Ind. Eng. Chem.*, III, 255) Use 0.5 gm. of the alloy, or more, according to the percentage of the constituents; remove Sn with HNO_3, Pb by H_2SO_4, Cu by electrolysis and Fe by NH_4OH. Add 5 gm. NH_4Cl, just neutralize with HCl, and add 0.4 gm. of dimethylglyoxime dissolved in a little alcohol for every 0.1 gm. Ni supposed to be present. Add NH_4OH drop by drop until there is a slight odor of NH_3. Let stand just below the boiling point for 30 minutes, or until the Ni separates completely. Filter hot through counterpoised filter or a weighed Gooch, wash with hot water, dry at 105° and weigh. The factor for Ni is 0.2031 deduced from the formula $NiC_8H_{14}N_4O_{11}$. It is well to test the filtrate with a little more of the reagent, making sure that the solution is ammoniacal, and allowing it to stand for a few minutes. If all has been separated, render the solution just acid with HCl, then add an excess of 10 c.c. and boil 10 minutes to destroy the dimethylglyoxime. The Zn can then be determined as phosphate in the usual manner.

IRON—STEEL—FERRO-ALLOYS.

Silicon in Irons and Alloys containing Vanadium or Molybdenum.—Trautmann. (*Z. angew. Chem.*, XXIV, 635) The SiO_2 obtained from such alloys invariably contains V or Mo compounds unless thoroughly washed with hot dilute HCl. When V is present the acid should be of 10-percent. strength; with Mo., a 5-percent. strength will suffice.

Carbon in Steel.—Isham. (*J. Ind. Eng. Chem.*, III, 577) An investigation of the question of loss of C on dissolving in K_2CuCl_4. (*vid.* QUARTERLY, XXX, 56) The results indicate that the apparent loss is

due to gaseous C compounds present in the reagent, and not to a loss of hydro-carbons from the steel itself.

Galvanized Iron.—Patrick and Walker. (*J. Ind. Eng. Chem.*, III, 239) The Preece test consists in a number of one-minute immersions in $CuSO_4$ solution of a certain strength under certain prescribed conditions. The number of immersions necessary to obtain a bright coating of Cu is taken as a gauge of the thickness of the Zn. The authors recommend the use of a slightly basic lead acetate solution: 400 gm. crystallized $Pb(C_2H_3O_2)_2$ in 1 litre, in which 4 gm. finely ground PbO is stirred until practically all has dissolved; the Sp. Gr. of the solution is then brought to 1.275 by dilution. The test piece is immersed for 3 minutes or more at a time, the precipitated Pb removed, and the operation continued until the Zn has been all removed and the Fe shows clean. The piece may then be weighed to determine the Zn by loss, or the weight of the Pb can be ascertained, and the Zn calculated. In the first case, if any Fe has been dissolved, the amount must be determined as a correction.

Nickel in Steel—Comparison of Methods.—Boyle. (*Chem. Eng.*, XIV, No 1) The dimethylglyoxime method, and the cyanide are compared. The conclusion is that the first is the more accurate, but that the cyanide method is quicker and very exact in the hands of one accustomed to it. The details are given thus:

Dimethylglyoxime.—Treat 1 gm. of the sample in a 250-c.c. beaker with 20 c.c. HCl (Gr. 1.12) keeping it covered. Heat until dissolved, then add 6 or 7 c.c. of HNO_3 (Gr. 1.2) to oxidize. Heat a short time to complete the oxidation, then add 20 gm. tartaric in 20 c.c. water, remove heat, and add cautiously strong KOH solution until solution is nearly neutral. The volume should now be about 200 c.c. Acidify slightly with acetic; heat to boiling, and add 15 to 20 c.c. of a 1-percent. solution of dimethylglyoxime in alcohol, with stirring. Then add NH_4OH carefully until the solution has a slight odor of NH_3. Filter while hot on a weighed Gooch, wash with warm water, dry at 110 to 120° and weigh. The precipitate is red and crystalline, sublimes at 250° without decomposition and contains 20.32 per cent. Ni.

Cyanide.—Dissolve 1 gm. in 20 c.c. of (1:1) HCl. Add 10 c.c. HNO_3 (Gr. 1.2), boil down to 15 c.c., keeping covered; then add 32 c.c. of 25-percent. H_2SO_4. Transfer to a 400-c.c. beaker, add 4 gm. anhydrous $Na_4P_2O_7$ and 5 gm. citric, stir, and slowly add NH_4OH until the precipitate just dissolves, and the solution has slight odor of NH_3. If too much NH_4OH has been added, nearly neutralize with dilute HNO_3. Dilute to about 150 c.c., cool to below 20° C., add 2 c.c. of a 10-percent. KI solution, and a measured amount of N/10 $AgNO_3$ in quantity sufficient to produce distinct turbidity. Titrate with standardized KCN solution, adding it slowly until the turbidity just disappears, and the solution shows yellow. Make correction for the $AgNO_3$ used, and calculate to Ni. The $AgNO_3$ solution should be exactly N/10, of which 1 c.c.=0.002935 gm. Ni. The KCN solution should be approximately N/10, but requires frequent re-standardization. The addition of 5 gm. KOH per litre improves its stability.

Vanadium in Iron and Steel—Colorimetric.—McCabe. (*Chem. Eng.*, XIII, 242) A chloride solution is obtained, oxidizers being used to bring the Fe to $FeCl_3$. An ether separation is then made, the V dissolving with the Fe in the ether. After expelling the ether, HF is added to destroy the coloration due to Fe, and then by use of H_2O_2, the depth of coloration due to V can be be estimated by comparison with that obtained from standard solutions.

Vanadium in Steels.—Cain. (*J. Ind. Eng. Chem.*, III, 476) Vanadium steels usually contain 0.1 to 1 percent. V. The lower figure is by no means negligible. The relatively wide differences obtained by different chemists on the same sample indicate error in many of the methods used. An investigation has shown that for titrating with permanganate (converting the V from the tetravalent to quinquivalent form) metals other than V must be removed from the solution. Incidentally, the observation of Hillebrand is confirmed, that the titration must be conducted at 70 to 80° C. to obtain a sharp end reaction. The method proposed consists in separating all the Cr and V from the most of the Fe by boiling with $CdCO_3$, the Fe being in the ferrous condition. After removing Cd by H_2S, the Cr and remaining Fe are separated by electrolysis, using a Hg cathode. The V remains dissolved, and is reduced and titrated. The details are:

Dissolve 2 to 4 gm. of the steel in 40 to 60 c.c. of 10-percent. H_2SO_4 in an Erlenmeyer. Fuse the insoluble with $KHSO_4$, dissolve, and add to the solution. Nearly neutralize with Na_2CO_3. Boil, and to the boiling solution add pulverized $CdCO_3$ at intervals until it is in excess to the extent of 1 or 2 gm. Settle and filter quickly to avoid reoxidation of much Fe. Wash twice with hot water. Dissolve back into the same flask with the minimum amount of hot 10-percent. H_2SO_4. Cool, nearly neutralize with NH_4OH, and precipitate the Cd by passing H_2S through the boiling solution. Concentrate, if necessary, to 60 or 70 c.c. and electrolize with Hg cathode, using 5 to 6 amp. at 6 to 7 volts. When the Fe is gone, as shown by side test with fericyanide, the Cr has usually also been removed. To test for Cr, try a little of the solution with H_2O_2, boil out the V color, and add NH_4OH. If the solution shows no color, Cr is absent. Draw off the solution, wash the Hg two or three times while the current is on, heat the solution and oxidize completely with permanganate; reduce with SO_2, expel the excess by CO_2, filter through asbestos, bring to 70 to 80° C. and titrate with N/50 permanganate.

Rarer Metals.

Titanium.—Muller. (*J. Am. Chem. Soc.*, XXXIII, 1506) The author notes the deep orange red coloration produced by adding salicylic acid to solutions of alkaline titanates. The color is destroyed or prevented by HF. The phenomenon was found to be applicable in a colorimetric estimation of Ti in presence of Ta. In presence of Cb or Th the results were somewhat high. Zr or Fe give strong colors of their own with the reagent, which precludes the use of the reaction under the conditions most frequently prevailing in analytical work.

Titanium.—Bornemann and Schirmeister. (*Metallurgie*, VII, 723)

Reduce the cold solution with $NaHSO_3$ and warm to 50° until KCNS will give no coloration for ferric. Then add a considerable excess of strong NH_4OH containing KCN and boil in a flask until the precipitate is white. Cool, protecting from oxidation of the Fe, and filter rapidly for the same reason. Redissolve and reprecipitate to purify from alkaline salts.

Separating Aluminum from Beryllium.—Wunder and Chiladze. (*Ann. Chim. anal.*, XVI, 205) A study of various methods has been made and the following is recommended: Get into solution by fusion with $KHSO_4$ and dissolving in dilute H_2SO_4. Precipitate by NH_4OH. Dissolve the precipitate in HCl and evaporate to dryness on the water bath. Add barely enough H_2O to moisten, and then add saturated NaOH or KOH drop by drop until both hydroxides are dissolved. Then add 400 c.c. of warm water, and boil for an hour, keeping up the bulk of the solution. $Be(OH)_2$ only is precipitated. Acidify the filtrate, and then precipitate out Al by NH_4OH. The method is applicable in presence of Fe.

Rare Earths—Separation.—Stoddart and Hill. (*J. Am. Chem. Soc.*, XXXIII, 1076) A hot alcohol solution of potassium stearate is added drop by drop to a cold neutral solution of the earth nitrates in sufficient amount to precipitate only a fraction at a time. A mechanical stirrer is worked in the liquid during the addition of the reagent. The precipitates are easily filtered and washed. A practically pure yttrium compound was obtained by this means from the earths of monazite, by comparatively few fractionations. The yttrium earth stearates are soluble in CCl_4, xylene, etc.

Cerium Separation.—James and Pratt. (*J. Am. Chem. Soc.*, XXXIII, 1326) $KBrO_3$ in neutral or faintly acid solution precipitates Ce on boiling, as basic nitrate with some basic bromate, free from the rare earths usually accompanying it in its minerals. To maintain the proper degree of neutrality in the solution, a lump of marble is added. Powdered marble, and several other substances tried, caused alkalinity of the solution with consequent deposition of Di. The more concentrated solutions gave the denser precipitates, which were the more easily washed. To prevent the formation of colloidal solutions it was found absolutely necessary to use a 5-percent. solution of NH_4NO_3 for washing. The operation was controlled by withdrawing small amounts of the solution from time to time and applying the H_2O_2 test. The presence of partially oxidized filter paper in the solution was found to be fatal to the success of the operation.

Cerium—Separation by Permanganate.—Roberts. (*Am. J. Sci.*, XXXI.) The separation from the other rare earths in nitrate solution can be made effective, provided some neutralizing agent is present to take up the HNO_3 set free. This is accomplished by use of a solution containing $KMnO_4$ and Na_2CO_3 in the proportion of 1 mol. (158.03 gm.) to 4 mol. (424 gm.) in any convenient amount of water. The solution (containing only nitrates) is brought to boiling in a porcelain dish, and, if acid, neutralized with pure Na_2CO_3 solution; then pure $KMnO_4$ solution is added until the color begins to be permanent. The

mixed solution is then added slowly, the boiling being maintained, a little more $KMnO_4$ being added if the solution becomes bleached. The faint pink color must be always present. Toward the end, the color bleaches out slowly. At the end the solution must be made absolutely neutral. To test the progress, the faint pink must be destroyed by alcohol, and a test made with NH_4OH and H_2O_2. $H_2C_2O_4$ can be used to precipitate the other earths from the filtrate. The precipitate is dissolved in strong HCl, the solution diluted and precipitated with $H_2C_2O_4$, converted to nitrate, and reprecipitated by the above process, to obtain a very pure oxide.

Thorium—Iodic Acid Separation.—Meyer. (*Zts. anorg. Chem.* LXXI, 65) The solution must be strongly acid with HNO_3, and also must contain a considerable excess of HIO_3. Two solutions are used: No. 1 contains 15 gm. KIO_3, 50 c.c. HNO_3 (Gr. 1.4) and 100 c.c. H_2O; No. 2 contains 4 gm. KIO_3, 100 c.c. HNO_3 (Gr. 1.2) and 400 c.c. H_2O. For qualitative test, add 5 c.c. of No. 1 to 2 c.c. of the Th solution (which should contain no chloride), then dilute with 10 c.c. of No. 2, and boil. As little as 0.0001 gm. Th may be thus detected in presence of the other earths. Zr gives a similar reaction, but the precipitate is soluble in $H_2C_2O_4$. The reaction is especially serviceable in separating Th from Sc. In quantitative work the $Th(IO_3)_4$, when obtained, is converted to $Th(OH)_4$ by boiling with NH_4OH, washed clean, and then weighed as ThO_2.

Vanadium in Ores.—Bleecker. (*Met. Chem. Eng.*, VIII, 666; IX, 209) Grind finely and fuse with Na_2O_2 in Ni crucible. Cool, leach out with water, and filter hot; V goes into solution. Acidify with H_2SO_4, heat to boiling, and reduce by adding 0.05 gm. $CuSO_4$ and electrolyzing. Titrate direct with permanganate.

Vanadium in Presence of Arsenic.—Trautmann. (*Zts. anal. Chem.*, L, 371) In many of the methods used for reduction of V, preliminary to titration with permanganate, As_2O_5 is also reduced and the AsO_3 destroys $KMnO_4$, thus vitiating the result. The author finds that As_2O_5 is not reduced by SO_2. To titrate for V only, when As is the only interfering substance, oxidize with $KMnO_4$, reduce with SO_2, expel excess of SO_2 by boiling, and then titrate the V only.

Palladium.—Gutbier and Falso. (*Zts. anal. Chem.*, XLIX, 287) From solutions of halogen salts of Pd, the metal can be precipitated as such by hydrazine. A necessary condition is that the Pd solution should be dilute, or, if a concentrated solution is used, both it and the reagent should be boiling when mingled. The metal first separates as a mirror on the sides of the vessel; a vigorous evolution of gas occurs, so that the vessel should be kept covered; the metal becomes detached, and breaks up into glistening scales. Heat for half an hour, let settle, and wash thoroughly by decantation. Ignite in a Rose crucible at first exposed to the air; then in a current of H and finally in a current of CO_2 free from O, in which it is cooled. The halogen which had been combined with the Pd can be determined in the filtrate.

Tungsten Concentrates.—Hutchin. (*Analyst,* XXXVI, 398) Description of methods in use.

(1) Aqua Regia.—Stir 1 gm. into 10 c.c. conc. HCl, then add 120 c.c. more, and boil (covered) down to about 5 c.c. Add 5 c.c. HNO_3 and digest hot for 5 to 10 minutes. Dilute to 100 c.c., settle and wash by decantation. Add 10 c.c. water, and 10 c.c. NH_4OH(1:1), washing the sides of the beaker down with the alkali. Stir and boil, repeating this until the insoluble settles readily. Filter through the original paper. Wash 3 or 4 times by decantation. Evaporate solution and washings, finally transferring to a weighed Pt dish; evaporate dry, ignite and weigh WO_3. Treat the insoluble in the beaker with 5 c.c. conc. HCl, evaporate low, then add a little HNO_3, and heat. Dilute and filter. Treat cover glass, beakers, filter, etc., with hot dilute NaOH. Collect all such washings, boil, add 2 gm. NH_4NO_3, stir and filter. Nearly neutralize with HNO_3 and precipitate recovered WO_3 with $HgNO_3$. Filter off, wash, ignite, and weigh recovered WO_3.

(2) Aqua Regia, Mercurous Nitrate.—Decompose as above, but instead of the NH_4OH treatment, add about 15 c.c. H_2O and 5 c.c. of 10-percent. NaOH to the residue, and wash beaker, cover glass and filter with more dilute NaOH. Add 3 gm. NH_4NO_3, boil, settle, filter, and wash by decantation with dilute NH_4NO_3. Dilute to 200 c.c., acidify with HNO_3. Then make neutral or slightly alkaline with NH_4OH. Add excess of $HgNO_3$ and neutralize again with HgO. Stir well, filter, and wash by decantation. Dry, ignite, and weigh as WO_3. The insoluble residue should be treated again to recover any WO_3.

(3) Soda Digestion.—Digest 0.625 gm. of the well pulverized sample with 20 c.c. of a 25-percent. NaOH solution in a porcelain dish for 45 minutes; dilute, add a little Na_2O_2 and transfer to a 250-c.c. flask, bringing the volume up to 250 c.c. Filter off 200 c.c. (=0.5 gm.). Acidify with HNO_3, then make alkaline with NH_4OH; boil, filter, and wash. Nearly neutralize the filtrate with HNO_3, and precipitate with $HgNO_3$, neutralizing with HgO as before. Filter, wash very thoroughly, and obtain WO_3 as described. Fine grinding, and very thorough washing of the Hg precipitate are the important points in this method.

(4) Soda Fusion.—Fuse 0.625 gm. in Ni crucible with about 1 in. of stick NaOH and 3 gm. Na_2O_2. Start the heating slowly, gradually increasing the intensity of the heat. Pour the melt out upon a clean sheet of Ni, and when cool dissolve in H_2O, leaching out the crucible as well. If the solution is green, add a little Na_2O_2. The rest of the operation is conducted as described in the soda digestion method.

Zirconia and Hydrofluoric Acid.—Wedekind. (*Ber.* XLIV, 1753) In expelling SiO_2 from admixture with ZrO_2 by HF, losses of Zr appeared to occur in some cases. Experiments showed that ZrF_4 was volatilized whenever the amount of H_2SO_4 used was insufficient. Using about 3 gm. HF on nearly 0.5 gm. of ZrO_2 alone, the loss was 13 percent.; with dilute H_2SO_4, the loss amounted to over 16 percent.; but on using 1.6 gm. conc. H_2SO_4, no loss was experienced. The observation was confirmed by testing mixtures of ZrO_2 with SiO_2. On the other hand in attempting to determine the SiO_2 in hyacinth, $ZrSiO_4$, the loss after four treatments was only 19.13 percent., whereas

it should have been 33.9 percent. The discrepancy was attributed to insufficient grinding of the mineral.

Molybdenum—Volumetric.—Knecht and Atack. (*Analyst,* XXXVI, 98) The Mo in strong HCl solution is reduced by Zn and then titrated with a standard solution of methylene blue. During the reduction and titration, all air must be excluded. This involves conducting the operations in a flask carrying a tube to supply CO_2 as the inert gas, a Bunsen valve tube, and the adapter for the burette. Through a rubber cork in the top of the Bunsen valve tube, is passed a Pt wire carrying a rod of Zn which can be lowered into the solution when necessary, to complete the reduction. The most of the reduction is effected by granulated Zn placed in the bottom of the flask, the solution (with 50 to 75 c.c. conc. HCl) being poured in upon it. The color of the solution passes through blue, green, orange, yellow, and finally pink, during the reduction. On titrating, the color changes are to yellow, greenish, and finally, with one drop of excess, the methylene blue color. $FeCl_2$ does not decolorize the methylene blue.

Thallium—New Reaction.—Tanatar and Petrov. (*Chem. Centrbl.,* 1910, I, 1291.) The reagent is $Na_3Co(NO_2)_6$ in acetic solution. A red precipitate of $Tl_3Co(NO_2)_6$ separates. Solubility in water is 1:10,000. Pb is the only other metal giving a precipitate resembling it, but the precipitate is yellow, and much more soluble.

Acids and Negative Ions

Chlorides, Chlorates and Perchlorates in Mixtures.—(*Am. Chem. anal.,* XVI, 167) Heating chlorates with HNO_3 affords nitrates. Perchlorates are not affected. Three portions of the sample are taken. In the first the chloride as such is determined in the ordinary manner. ($AgNO_3$ titration, or similar method). In the second, total Cl is determined by fusion with six times its weight of Na_2CO_3 and KNO_3. Dissolve in dilute HNO_3, neutralize, and titrate total Cl. The third portion is repeatedly evaporated with HNO_3 (36 to 40° Be.) on a sand bath until the solution gives no cloud with $AgNO_3$. Then fuse with Na_2CO_3 and KNO_3 as before, and obtain Cl of the perchlorate as chloride, which can be titrated. Cl as chlorate is obtained by difference.

Bromine—Qualitative.—Deniges. (*Bull. Soc. Chim.,* IX, 542) To a 1-percent. solution of strychina sulfate add 5 c.c. pure HCl and 4 or 5 gm. granulated Zn. Heat to boiling, then let stand a short time, and decant the solution. When freshly prepared, 1 c.c. of this reagent, when shaken with a solution containing Br, develops a characteristic purple coloration. The limit of sensitiveness is about 0.01 mg. Br in 1 c.c. of liquid. The reagent gives a red color with nitrate.

Bromine in Potassium Chlorate.—Pieszczek. (*Pharm. Ztg.,* LIV, 325) 20 gm. of the sample were mixed with an equal weight of freshly precipitated $Fe(OH)_3$ and heated to 120°. O was evolved and the halides were thus obtained. The residue was dissolved in water, filtered, the filtrate evaporated by dryness, and extracted with strong alcohol, which dissolved all the KBr, and but little KCl; filtration and evaporation of the alcoholic solution followed. The residue

was taken up with water, affording a solution in which the ratio of Br to Cl rendered the more usual methods of separation available. The method pursued by the author was precipitation of the Ag salts, weighing, treating with dry Cl, and weighing again.

Sulfur in Coal.—Warunis. (*Z. anal. Chem.*, L, 270) Mix 1 gm. of the coal with 2 gm. CuO and 1 gm. Na_2CO_3. Ignite in porcelain crucible with constant stirring until all C is burned off. This takes about 2 hours. Cool, add Br water, dissolve in HCl, filter, and precipitate boiling with $BaCl_2$ and weigh in the usual manner.

Sulfur in Roasted Pyrites.—V. Hohorst. (*Chimiste,* II, 97) Mix 1 gm. of the dry sample with 2.5 gm. of a mixture of powdered Al and Fe (1:3), and load the mixture into a porcelain tube. A cover of the Al-Fe mixture is added. The tube is tapped gently when in a horizontal position, to give a passage for the gases evolved. It is then gradually heated to redness. After cooling, the mass is treated in a flask with strong $SnCl_2$ solution, then 30 c.c. of 1:1 HCl is added and a current of air is passed through the flask, the escaping gases, containing the H_2S, being passed through two wash bottles containing $Cd(C_2H_3O_2)_2$. The rest of the operation is conducted in the ordinary manner for S by evolution methods in iron and steel analyses.

Sulfates—Jackson's Candle Turbidimeter.—Muer. (*J. Ind. Eng. Chem.*, III, 553) (*vid.* QUARTERLY, XXIII, 288) The process can be carried through in 10 minutes on the washings from the bomb calorimeter, in the case of coals. An investigation was made as to the reasons for certain anomalous results; the conclusions were:

(1) Variations in the size of the candle flame, or its distance from the observation tube, have no appreciable effect.

(2) Nitrates, to the extent usually occurring, have no effect.

(3) A large excess of HCl gives low results with weak sulfate solutions. Acidification should not be effected until the test solutions are diluted to nearly 100 c.c.

(4) The principal cause of the differences noted, was due to the size and hardness of the $BaCl_2$ crystals added. This difficulty was met by using compressed tablets of $BaCl_2$ (1 gm. each) as the precipitant.

(5) Applying the tests in the manner described renders the method convenient and accurate, as applied to coals, cements and boiler waters. In the case of the latter, the water should contain at least 37 parts SO_3 per million, or over. If weaker in sulfates, the water should be concentrated before using the test.

Sulfides in Alkaline Cyanides.—Rossiter. (*J. S. C. I.*, XXX, 583) Three methods were examined. In all cases the water used for dissolving the cyanide samples was freed from air by boiling, and cooled without access of air.

Hg Method.—Dissolve 10 gm. of the sample in 50 c.c. H_2O and add 5 c.c. of a saturated solution of $HgCl_2$. The author recommends warming to coagulate the HgS, then cooling, and rendering barely acid with HCl before filtering. Filter on a weighed filter, dry at 110° and weigh HgS. Acidification with HCl before filtering was found to obviate the necessity for applying a correction, otherwise found to be

necessary because of some solvent action of the cyanide on the filter paper.

Williams Method.—Dissolve two portions of 5 gm. each in separate lots of 100 c.c. of water. To one lot, add 0.5 gm. $PbCO_3$, shake and filter. This solution is used as a comparison solution. Put the solutions in Nessler tubes, and to each add 1 c.c. of Na_2PbO_2 (made by digesting PbO in 5-percent. NaOH). To the comparison solution add standard Na_2S solution until the colors match. The author finds that working in this manner gives a different quality of tint (colloidal PbS in one case, and not in the other). He recommends to make a preliminary test in this way, then to prepare two other fresh solutions, and to add to the comparison solution nearly enough Na_2S to give the tint, and then to add the Na_2PbO_2 to both, finishing the titration by adding the necessary amount of Na_2S to match the colors.

Ewan Method.—Stir 10 gm. of the sample into 15 c.c. of H_2O until nearly all is dissolved, and then titrate with $Pb(NO_3)_2$ solution, until a drop on thick blotting paper saturated with $Pb(NO_3)_2$ solution gives no brown tint. Calculating by molecular weights, 3 gm. $Pb(NO_3)_2$ per litre should give a solution of which 1 c.c.=0.01 percent. K_2S or 0.0071 Na_2S. Ewan advises the use of 2.4 gm. of the salt to allow a 20-percent. correction for oxidation. The author finds that nitrates favor oxidation of the sulfide, but the correction should be about 10 percent. The author recommends a preliminary test by this method, then to weigh out 100 gm. of the sample, and to add a known amount of pulverized dry $Pb(NO_3)_2$ in quantity sufficient to be 10 percent. in excess of that required to precipitate all the S. After filtration, the excess of Pb is titrated with standard Na_2S until a few drops of the solution, on filtering, shows no trace of Pb by Na_2S.

Volumetric for Phosphoric Acid.—Rosen. (*J. Am. Chem.*, XXXIII, 1099) Chlorides must be excluded from the solutions and reagents used. The solution containing the phosphate is made neutral to phenolphthalein, and a known amount of N/10 $AgNO_3$ is added. After thorough shaking, ZnO is added until the solution is neutral to litmus. A large excess of ZnO, or long standing, is to be avoided. Filter, and in an aliquot portion of the solution determine the Ag remaining by Volhard's method. The Ag taken by the P_2O_5 is reckoned as Ag_3PO_4, and the calculation is made accordingly.

Phosphates—Colorimetric Method.—Passerini. (*Gaz. chim. ital.*, XLI, 182) The method is based on the coloration caused by adding gallic acid to a neutral solution of ammonium-phospho-molybdate. (Yellow, orange or red according to the quantity of phospho-molybdate.) Very detailed directions are given for obtaining the phospho-molybdate from the phosphate of the sample to be tested. They do not, however, differ materially from the ordinary line of treatment. Organic matter in the sample must be destroyed by ignition before dissolving. The yellow precipitate, when obtained, is dissolved in the least possible quantity of NH_4OH, the NH_3 boiled out, and the solution brought to 100 c.c., which is run from a burette into a compari-

son tube containing 15 c.c. of water and 2 c.c. of a saturated solution of gallic acid, until the color matches that of the standard.

The standard is prepared by dissolving 1.9153 gm. pure $CaHPO_4$, obtaining the phospho-molybdate therefrom, dissolving in NH_4OH as above, and diluting to 1 litre; 1 c.c.=0.00001 P_2O_5. For comparisons, 1, 1.5 or at most 2 c.c. of this solution is used in the comparison tube with water and gallic acid, as described.

Fluorides.—Starck. (*Zts. anorg. Chem.,* LXX, 173) After neutralizing the solution, an excess of saturated solution of $PbCl_2$ is added; for 0.1 gm. NaF use 200 c.c. of the $PbCl_2$ solution. The precipitate is PbFCl. Allow to stand over night, filter through a Gooch, wash with saturated $PbCl_2$ solution, then 2 or 3 times with water, dry 2 hours at 140 to 150° and weigh. The degrees of solubility of PbFCl in $PbCl_2$, in H_2O, in HCl and in acetic are recorded. The stronger the $PbCl_2$ solution the lower the solubility. Under the conditions specified, it is practically nil at 18°. 100 c.c. H_2O dissolves 0.0325 gm. at 18° and 0.037 gm. at 25°, and much more at higher temperatures. Weak HCl or acetic have a still stronger solvent action.

Potassium Cyanide.—Treadwell. (*Zts. anorg. Chem.,* LXXI, 219) The $AgNO_3$ method is inaccurate in presence of ferrocyanide, which has a solvent effect on AgCN. It is found that if the KCN solution is not stronger than N/10, and is rendered slightly alkaline with NH_4OH, and 1 gm. KI be added, an accurate titration can be obtained. $K_2S_2O_3$, if present, is also prevented from interfering by this means, if the proportion is not too large.

Calcium Cyanamide.—Brioux. (*Ann. Chim. anal.,* XV, 341) Dicyanamide is considered by the author as injurious to plant life. Its determination is therefore of great importance. The method applied is: Treat 1 gm. with water in a 250-c.c. flask with frequent shaking for 3 hours; dilute to the mark. Filter and take 100 c.c. of the filtrate. Add 20 c.c. of 5-percent. $AgNO_3$, then excess of NH_4OH. Filter off the yellow precipitate, wash with dilute NH_4OH, then with water; dissolve in warm dilute HNO_3 and titrate with N/10 NH_4CNS. This gives N as cyanamide; 1 c.c.=0.0014 N. To another 100 c.c., add $AgNO_3$ as before, and also 20 c.c. of 10-percent. KOH, filter, wash and determine N by Kjeldahl process, adding about 1 gm. $CuSO_4$ and the usual amounts of conc. H_2SO_4. This gives the sum of N in both cyanamide and dicyanamide. Subtract the N found by the first process; the remainder is N of the dicyanamide.

Calcium Cyanamide—Lime-Nitrogen.—Monnier. (*Chem. Ztg.,* XXXV, 601) The method consists in converting the cyanamide to urea, and determining by means of hypobromite. Treat 1 gm. of the sample for 2½ hours with 100 c.c. of water, agitating frequently. Filter, and evaporate 25 c.c. of the filtrate with 10 c.c. formic acid nearly to dryness. Add water enough to bring the volume up to 25 c.c. and test 10 c.c. with NaBrO in a ureometer, making a comparison test with a 0.5-percent. solution of urea. The Ag method (*vid.* QUARTERLY, XXXII, 405) is asserted to give low results, the Ag compound being somewhat soluble. The commercial product contains some un-

identified compound of N which is easily oxidized to nitrate by treating it with H_2O_2, and can in this way be determined. Stutzer (*ib.* 694) disputes the above conclusions as to the Ag process.

Ferrocyanide.—Schröder. (*Zts. anorg. Chem.,* LXXII, 89) A method of analysis is mentioned, consisting of decomposing by $HgCl_2$ and then titrating the Fe with permanganate.

Nitrogen in Nitrates.—Quartaroli. (*Chem. Centr.,* II, 49) Anhydrous formic acid, when heated with nitrates, acts thus:

$$2KNO_3+6HCHO_2=N_2O+4CO_2+5H_2O+2KCHO_2$$

By working with 0.1 to 0.3 gm., and collecting the gases over Hg in a eudiometer, the N and, *per contra,* the nitrate may be determined. The author reads off the volume of mixed gases, (N_2O and CO_2) and then absorbs the CO_2 by KOH and reads again as a check.

MISCELLANEOUS

Peroxidized Compounds.—Lenz and Ricther. (*Zts. anal. Chem.,* L, 537.) NaBO with $K_2Cr_2O_7$ and a little ether, gives the blue of perchromic acid. The reaction may be used to distinguish an alkaline borate from a perborate. If to a 1-percent. solution of a per salt, one adds a couple of drops of 10-percent. KI, I is liberated only if a persulfate is present. Acidification with dilute H_2SO_4 will then give free I if a perborate or percarbonate is present, but not with perchlorate. With these substances, under the same conditions, no Br is liberated from KBr.

Lead Arsenate for Plant Spraying.—Volck. (*Science,* XXXIII, 866) The commercial forms of lead arsenate frequently contain $PbH(AsO_4)_2$ or $Pb_2As_2O_7$, which, by reaction with the alkaline constituents of the water used for dilution or with ammoniacal emanations from fertilized soil, etc., afford $Pb_3(AsO_4)_2$ and a soluble arsenate. The soluble arsenate, even in small proportions, is destructive to the foliage. Only the $Pb_3(AsO_4)_2$ should be used. The test adopted for detection of the objectionable compounds consists in stirring 10 to 20 gm. of the preparation into 25 to 50 c.c. of water, adding an equal bulk of NH_4OH solution, and heating slowly to the boiling point, agitating frequently meantime. After settling and cooling the clear liquid is decanted through a filter, and the solution boiled until NH_3 has been expelled. Then add a strong solution of lead acetate containing free acetic. If no precipitate forms at first, add more of the Pb solution; stir in and allow to stand for an hour or so. The formation of a precipitate or of more than a slight opalescence indicates the presence of the objectionable forms of lead arsenate, and the material is unsuited for spraying purposes.

Calorimetry—Anthracite Coal.—Blakeley and Chance. (*J. Ind. Eng. Chem.,* III, 557) The difficulty with anthracite—a portion remaining unburned—has long been a source of annoyance. The authors avoid it by packing a little fine, thoroughly ignited, asbestos into the nickel cup, and resting the coal upon that. The mass being thus thermally insulated from the Ni cup and heavy Pt wire, is completely burned.

Formic Acid—Delicate Test.—Deniges. (*Pharm. J.,* LXXXVI, 773)

A solution containing one part of methylene blue in 5000 is prepared. Of this, add in the proportion of one drop for each c.c. of the formic solution, when of 1-percent. strength or less. (e.g. 5 c.c. requires 5 drops; if the acid is stronger, increase proportionately.) Boil, and immediately add the same number of drops of $NaHSO_3$ solution of 36 to 40° Be. Decolorization ensues on shaking, if formic acid is present.

Naphthalene in Gas.—Albrecht and Müller. (*J. f. Gasbeleuch,* LIV, 592) To remove tar, CN, H_2S, and NH_3, the gas must first be passed through a couple of wash bottles containing dilute H_2SO_4, and then through a couple containing dilute KOH. Then, to collect and remove the naphthalene, it is passed through two wash bottles, each of 10 c.c. capacity, containing 2.5 gm. picric acid, distributed approximately evenly between them. Each flask contains only 25 c.c. of water, so that the picric is only partially in solution. The connections in the train should be made with butted glass tubes, that the rubber tube holding them together shall not be in contact with the gas. The gas is passed at the rate of 40 or 50 litres per hour, until it is judged that 0.05 to 0.2 gm. of naphthalene has been taken up. The contents of the wash bottles are then united in a 250-c.c. flask, and the picric brought into solution by addition of water and warming, the solution being eventually made up to 250 c.c. when cold. 200 c.c. are then filtered off, and titrated with N/10 KOH and lacmoid indicator. 2.5 gm. of the picric used is also dissolved, and titrated in the same manner. The difference in the c.c. used is multiplied by $5/4 \times 0.0128$ to give grams of naphthalene. A reading of the meter shows the proportion in the gas.

Graphite in Lubricants.—Marcusson and Meyerhelm. (*Chem. Ztg.,* XXXV, 461) Treatment with C_6H_6 may serve when only lubricating oils or greases have been used in the mixture. If lime soaps have been used, dissolve off the fat and soap with a mixture of 90 parts C_6H_6 and 10 parts alcohol. The residue should be treated with HCl to leave the graphite. If the mixture contains tars or pitches, extract with C_6H_6 and add the residue gradually to NaOH in a state of fusion in a Ni crucible; cool, leach out with water and HCl, and weigh graphite dried at 105°. The method is not absolutely exact, but is practically serviceable.

Lubricating oils.—Marcusson. (*Chem. Ztg.,* XXXV, 729 and 742) The constituents in mineral lubricating oils which are of value do not react with H_2SO_4 and formaldehyde affording "formolite." Dissolve 27 gm. of the oil in 50 c.c. of benzine, and add (without shaking) 30 c.c. conc. H_2SO_4. Cool in ice water, and add 15 c.c. of 40-percent. formaldehyde; shake and cool until there is no further reaction (indicated by heating). Let stand 30 minutes, and then pour into a flask containing 200 c.c. of ice water; neutralize with NH_4OH and filter. Wash the precipitate free from oil by the petroleum spirit, then wash free of NH_4OH by H_2O. Dry to constant weight at 105° C. The petroleum spirit should be evaporated off, and the residual oils weighed.

Tarry Matters in Mineral Cylinder Oils.—Holde and Meyerheim. (*Chem. Ztg.,* XXXV, 369) Dissolve 5 gm. of the sample in 25 times its volume of ether, and add 12½ volumes of 96-percent. alcohol drop by

drop. After shaking, and then standing for 5 hours at 15° C., the solution is filtered and washed clean with the alcohol-ether solution. The residue, consisting of tarry matters and paraffin, is dissolved in benzol into a weighed dish, from which the benzol is evaporated, and after drying 10 minutes at 105° C. the tar and paraffin are weighed together. This mass is dissolved in 10 c.c. of ether, and poured onto a mixture of 10 to 15 gm. of sand mixed with 2 gm. bone black. Then extraction with absolute alcohol removes only the paraffin. This last extraction is accomplished in an apparatus specially constructed with ground joints so as to eliminate any cork connections. Loebell (*Petroleum,* VI, 774) describes essentially the same process, the proportion of the alcohol-ether mixture being somewhat larger, and cotton wool, instead of bone black and sand, being used to hold the tarry or asphaltic matters.

Oils for Automobiles and Gas Engines.—Schlüter. (*Chem. Ztg.,* XXXV, 413) The tests ordinarily applied do not show whether an oil will leave any residue in the cylinder or impart a bad odor to the exhaust. A determination of the proportion of the oil insoluble in acetone will serve roughly to determine this point, the best oils in this respect being the more insoluble in that menstruum.

Rosin in Admixture with Coal Tar Residues.—Holde and Meister (*Chem. Ztg.,* XXXV, 793) Mixtures of the kind indicated in the title are sometimes used as substitutes for asphalt. To determine the rosin approximately, 10 to 20 gm. of the material (less, if the rosin is 4 percent. or over) is extracted with ether under a reversed condenser. The ethereal solution is repeatedly shaken with successive lots of N/10 NaOH (30 c.c. each time) until the aqueous layer appears colorless. The saponified solutions are shaken with more ether, and the united ethereal extracts are shaken with a fresh lot of N/10 NaOH. The united alkaline solutions are treated with H_2SO_4 in presence of ether, some Na_2SO_4 added, and the ether washed. After treating with a little bone black, the ether is evaporated off and the rosin weighed after drying 5 minutes at 105°; 8 percent. of the weight is added to allow for the proportion of rosin not dissolved by the NaOH.

Cement—Rapid Analysis.—Wiley and Ernst. (*J. Ind. Eng. Chem.,* III, 551) By using Hentschel's acetic method for separating SiO_2, the time required for the analysis of a Portland cement (excluding alkalies) can be reduced to 2½ hours. The method is: Treat 0.5 gm. in a casserole with 10 c.c. glacial acetic, and a little water. Stir and break up all lumps with a rod; digest on the hot plate for about 5 minutes, stirring occasionally. Add dilute HCl until the red color is destroyed. Filter off SiO_2, wash, ignite, blast and weigh. Make the filtrate up to 200 c.c. and divide in halves. To the first add a few drops of HNO_3, and then precipitate Fe_2O_3 and Al_2O_3 by NH_4OH. In the filtrate, diluted to 400 c.c., Ca is precipitated as CaC_2O_4; wash, and titrate with permanganate. Concentrate the filtrate to 200 c.c., acidify with HCl, rinse into an Erlenmeyer, add $NaNH_4HPO_4$, and about 5 c.c. NH_4OH, cool, shaking under the tap, until the precipitate begins to form. Then add 15 c.c. NH_4OH, shake 10 minutes, settle for 5 minutes, and filter, etc. In the second half determine Fe by reducing with $SnCl_2$, adding it drop by drop to the boiling solution, cool, make up to 300 c.c. add 25 c.c.

$MnSO_4$ solution. Stir well, add 10 c.c. saturated $HgCl_2$ solution and titrate with permanganate.

The $SnCl_2$ solution contains 25 gm. $SnCl_2$ dissolved in 100 c.c. conc. HCl and diluted to one litre. The $MnSO_4$ solution is made by dissolving 67 gm. $MnSO_4$ crystals in about half a litre, adding 138 c.c. H_3PO_4 (Gr. 1.70) and 130 c.c. conc. H_2SO_4, and diluting the whole to one litre.

Meantime, for the SO_3 determination, 1 gm. is treated with acetic as before, and in the filtrate from the SiO_2, precipitation with $BaCl_2$ is effected, a few c.c. of alcohol being added, to assist the rapid settling of the precipitate.

Potassium by Cobaltinitrite Method.—Thompson and Morgan. (*J. Ind. Eng. Chem.*, III, 398) A modification of Adie and Wood's method is described as the form adopted in examination of ashes of cereals. The reagent is made by dissolving 113 gm. $Co(C_2H_3O_2)_2$ in 400 c.c. 20-percent. $HC_2H_3O_2$ and 220 gm. $NaNO_2$ in 400 c.c. of water, then mixing the two solutions, and allowing to stand for 24 hours; the NO formed is removed by use of a filter pump. For the titration, as 1 c.c. of N/10 normal $KMnO_4$=0.000785 K_2O, it is more convenient to make the strength 0.1274 normal, so that 1 c.c.=0.001 K_2O. The HCl solution of 0.5 gm. of ash is neutralized with NaOH, and then acidified with acetic, adding 10 c.c. of 50 per cent. acid in excess. Boil to precipitate Fe and Al phosphates, filter and wash. Concentrate to about 75 c.c., cool, add 25 c.c. of the nitrite reagent, and allow to stand over night. Filter through asbestos in a Gooch, wash several times with cold 10 percent. acetic, then once with water. Then transfer asbestos and precipitate to a small beaker, add 40 c.c. of saturated $Ba(OH)_2$ solution, and boil. Let settle, and filter hot into a 200-c.c. flask, washing with hot water. Cool, and make up to 200 c.c.

Run 25 c.c. of the standard permanganate into a casserole, add 5 c.c. of 1:1 H_2SO_4, and 150 c.c. of hot water, and run the nitrite solution into it slowly from a burette, until the color is just discharged. The use of $Ba(OH)_2$, instead of NaOH, was found to be a decided improvement; with large amounts of NaOH, the determinations proved very inaccurate.

Sulfur Dioxide in Gelatines.—Leffmann and La Wall. (*Analyst*, XXXVI, 271) The method consists in dissolving in water, adding H_3PO_4, and distilling in a current of CO_2, the distillate being condensed in a solution of I. From this, the S is precipitated out as $BaSO_4$. Many manufacturers are able to keep the proportion of SO_2 down to 50 or even 10 to 15 parts per million. Two brands, specially imported for use by bacteriologists, were found to contain respectively 265 and 835 parts per million—amounts likely to interfere seriously with bacteriological work.

Boric Acid in Preservatives.—V. Fellenberg. (*Chem. Centr.*, 1911, I, 1248) Dissolve 1 gm. of the sample in 2 c.c. of hot water, and add 1 to 2 c.c. conc. HCl. Cool; the deposit will consist of H_3BO_3 with alkaline chloride. Decant the supernatant liquid as closely as possible, and boil the residue with 4 to 6 c.c. alcohol. Acidify with H_2SO_4, and set fire to the alcohol.

Tin in Canned Foods.—Schreiber and Taber. (U. S. Dept. Agric.,

Chem. Circ., 107) A new plan of operation is described with much minuteness of detail which cannot be given here. 100 gm. of the material, with 10 gm. MgO, are weighed out in an 8-oz. wrought-iron crucible, and 50 c.c. of a solution containing 150 gm. NaOH and 100 gm. Na_2CO_3 per litre is added; after stirring, 75 c.c. of 95-percent. alcohol is stirred in. After this, the mixture is evaporated to dryness with constant attention to avoid loss by foaming. Eventually, the carbon is burned off by careful heating in a muffle, starting with it cold. After burning off, the contents of the crucible are leached out with water, and acidified with H_2SO_4. More H_2SO_4 and a little HNO_3 are added, and the solution evaporated to fumes. After dilution, with the necessary precautions, H_2S is passed. The precipitate is dissolved in KOH, and reprecipitated by HCl. The SnS_2 is finally ignited to SnO_2 for weighing.

Nitrates and Nitrates in Water.—Deniges. (*Bull. Soc. Chem.*, IX, 544) The reagent is made by treating 5 gm. Zn amalgam with a mixture of 5 c.c. HCl (Gr. 1.18) and 5 c.c. of 1-percent. solution of strychnia sulfate. Boil, cool and decant. 0.5 c.c. of the reagent is added to 10 c.c. of the water to be tested. Nitrites, if present, give a red coloration at once, and can be colorimetrically estimated. If 5 c.c. of H_2SO_4 is added, a coloration due to nitrates appears, which can be estimated in the same manner. If it is desired to estimate nitrates direct, add to 10 c.c. of the water 2 drops of NH_4OH, and 3 or 4 drops of acetic, and evaporate to dryness to eliminate nitrites. A repetition of this treatment may be desirable to effect complete removal. Finally, take up with 10 c.c. of water and test for nitrates as above.

Nitric and Nitrous Acids in Waters.—Tillmans and Sutthof. (*Zts. anal. Chem.*, L, 473) The diphenylamine test is used (*cf.* QUARTERLY, XXXII, 402) The reagent is prepared by treating 0.085 gm. diphenylamine with 190 c.c. of dilute H_2SO_4 (1:3), and then gradually adding conc. H_2SO_4 to bring the bulk up to 500 c.c. in the cold. To 100 c.c. of the water to be tested, add 2 c.c. saturated solution of NaCl. 1 c.c. of this mixture is then put into a comparison tube and 4 c.c. of the diphenylamine reagent carefully added. After shaking well, the tube is set aside for an hour, when the blue tint will have reached its maximum. It lasts for one or two hours, and then slowly fades. Standards for comparison are made by adding 0.5, 1, 1.5., c.c. of KNO_2 solution to 100 c.c. lots of water, as well as 2 c.c. saturated solution of NaCl to each, and taking out 1 c.c. of each for the test. The nitrous acid test is more sensitive, and the reagent can be made by diluting 500 c.c. of the above described reagent with 200 c.c. of water. The test is made in a manner similar to the above, but 5 c.c. of the water, and 5 c.c. of the reagent are used; the maximum color is attained in 10 minutes, and holds only for half an hour.

To get the nitrate only, the nitrite may be destroyed by adding to 100 c.c. 5 drops H_2SO_4 and 0.2 gm. urea and standing over night; or by evaporating down to 25 c.c. with 5 c.c. of 10-percent. NH_4Cl and diluting to the original volume (100 c.c.). For standards with 0.1872 gm. KNO_3 per litre, 1 c.c.=0.1 mgm. N_2O_5, and with 0.1816 gm. KNO_2 per litre, 1 c.c.=0.1 mgm. N_2O_3.

DEPARTMENTAL REPORTS.

DEPARTMENT OF CHEMISTRY.

(*Extracts from annual report of head of department for year* 1910-11).

General—Charles F. Chandler retired in June, 1911, after 55 years of active work, 47 of which were spent at Columbia.

Professor Alexander Smith, of the University of Chicago, has been appointed by the Trustees to take charge of the work in general and inorganic chemistry, and also to act as administrative head of the department. He assumed his duties on July 1, 1911.

Drs. Metzger and Beans and Mr. Hall have been promoted to assistant professorships.

Prof. Sherman's work has been set apart from analytical chemistry under the title "Food Chemistry."

The title "Industrial Chemistry" has been changed to "Engineering Chemistry" as more in accord with the best practice at the present time, and also for the reason that the students in this course receive the degree of Chemical Engineer.

The department will be well represented at the 8th International Congress of Applied Chemistry, which is to be held in this country in September, 1912, by special act of Congress and the invitation of President Taft. William H. Nichols, honorary D. Sc., of Columbia, is president of the congress and chairman of its executive committee; Prof. Bogert is vice-chairman of the executive committee, chairman of the committee of presidents of sections, and president of the organic section; Prof. Whitaker is a member of the executive committee; Prof. Smith is vice-president of the section of physical chemistry; Prof. Sherman is a member of the general committee of the section of agricultural chemistry; Prof. Tucker is a member of the executive committee of the section of electrochemistry; Mr. H. L. Fisher is secretary of the organic section.

The number of students taking laboratory courses during the current year has been the greatest in our history. Several of the laboratories have had many more students than they could possibly accommodate, and the overflow has been cared for as well as possible in other laboratories, although at considerable disadvantage both to students and to teachers. The removal of the Department of Architecture from the top floor of Havemeyer, and the turning over of these rooms to our department, will afford temporary relief, but at the present rate of growth this additional space will soon be fully occupied.

The recent establishment at Columbia of the honor chemical society, Phi Lambda Upsilon, has already proved of considerable value to this department. It provides an added stimulus to good work in chemistry since only the best students are admitted to membership. Further, it makes possible to bring together, in a most enjoyable social way, these students with the pick of our graduates and the members of

the teaching staff, and is thus a valuable adjunct in the extension of the influence of the department.

Analytical Laboratory.—A number of researches have been in progress during the year, several of which are now in preparation for publication. F. W. Zone, candidate for the degree of Ph. D., has completed his work on "A New Volumetric Method for the Determination of Chlorine and its Application to the Analysis of Monazite." L. E. Marrs, candidate for the degree of A. M., has just completed an investigation on "A New Rapid and Accurate Volumetric Method for the Determination of Manganese, and its Application to theAnalysis of Iron and Steel." A few of the investigations now under way are as follows: "A New Volumetric Method for the Determination of Lead;" "A New Colorimetric Method for the Determination of Cerium;" "The Electrolysis of Salts from Non-aqueous Solvents."

Attendance on the Department of Chemistry for the year 1910-11 is shown in the following table:

STUDENTS TAKING ONE OR MORE SUBJECTS IN CHEMISTRY. 1910-11.

Pure Science, A.M.	27
Pure Science Ph.D.	29
Extension Chemistry	61
Teachers College	4
	121

Applied	Science,	1911	37	Columbia	College,	1911	18
"	"	1912	84	"	"	1912	26
"	"	1913	183	"	"	1913	55
"	"	1914	191	"	"	1914	116
"	"	1915	2	"	"	Specials	4
"	"	Specials	4				219
			501		Total		841

The total registrations of all students in all courses during the year 1910-11 amounted to 2184, as compared with 2057 during the preceding year. M. T. B.

DEPARTMENT OF CIVIL ENGINEERING.

(*Extracts from annual report of head of department for year* 1910-11).

The first two years of the course have remained unchanged in all important particulars. The subject of roads, streets, and pavements, hitherto given by an instructor has been transferred to the junior Professor of Civil Engineering, in whose hands it is now administered in connection with the general subject of railroad engineering. The field of highway engineering is more or less related to that of railroads, and it is believed that the change will contribute both to the efficiency of administration and instruction.

The year just closing is the second during which the entire lecture and classroom work in elasticity and resistance of materials has been given in the first semester, the schedule being five hours per week for that session. This subject is fundamental to much of the subsequent instruction in all the engineering courses, and it is gratifying to find by actual experience that the present method gives more continuity and

efficiency in instruction than the former schedule of three hours a week in the first semester and two in the second.

The conduct of the fourth-year subjects, including the theses, has been characterized by a degree of efficiency at least equal to that of the past, and I believe with such enhancement of efficiency as we constantly aim to attain from year to year. The work performed by the students has, on the whole, certainly ben equal in quality to that of the past. The number of deficiencies in examinations has been somewhat less this year than in the preceding.

The most marked expansion in the department is the subject of highway engineering. The inception of this course is due to Professor Arthur H. Blanchard, recently of Brown University, Providence, R. I., who some years ago pursued graduate work in this department. Professor Blanchard secured from Mr. Charles H. Davis, of South Yarmouth, Mass., a graduate of the School of Mines of Columbia University in 1887, a gift of $10,000 per year for a period of five years, to be devoted to the administration and maintenance of an advanced or graduate course in highway engineering. This gift has been accepted by the Trustees of the University, and Professor Blanchard began his services about the first of November, as Professor of Highway Engineering, his appointment having already been made by the Trustees. Another instructor is to be appointed to aid Professor Blanchard in this work. This course will consist of two sessions, each beginning about November first and continuing until about April first. Those graduate students who complete this course will be qualified for the master's degree. The session is placed during the winter so as to enable many young engineers engaged in highway work to take advantage of the suspension of their field operations during the cold weather to take this educational work. It is believed that this course will afford such instruction as will place those who take it on the highest plane of professional efficiency in this particular field. In doing this, the University will render a great public service, raising the highways of the country, it is hoped, to the same class in which the best highways of Europe are found. In fact, it is the aim of this extension of the department to afford instruction equivalent to that given by the best French or other engineering schools of Europe.

I cannot close this report without expressing great gratification at the adoption by the faculty of the report of the committee on instruction, recommending six years as the full professional course of engineering study at Columbia University. As you are aware, it is a matter which I have strongly advocated ever since my connection with the University, and all interested in the welfare of the engineering school may look forward with confidence to its suitable development. I desire to emphasize what I wrote in connection with this subject in my annual report of a year ago, and will quote from it the following paragraph: "The course in civil engineering, as originally laid out seventeen years ago, and as now administered in Columbia University, is intended to fulfil the requirements of such a well rounded and complete professional training, and neither it nor any similar course can ever reach its full

effectiveness until it can be administered under the favoring conditions of a completely developed professional school, nor can any such course of educational training reach its full University significance under any other condition." W. H. B.

DEPARTMENT OF ELECTRICAL ENGINEERING.

(*Extracts from annual report of head of department for year* 1910-11).

The department of electrical engineering has gone through a state of transition during the past year, owing to the continued absence on leave of its former executive, Prof. Crocker, and the fact that Prof. Sever has also been absent on leave. Thus, with the coming of the undersigned, a rearrangement of courses and duties has been necessary. This readjustment of duties has been arranged very satisfactorily to all concerned, due to the praiseworthy spirit of coöperation and willingness among the members of the department. It has been customary throughout the year to hold departmental meetings weekly, at which every active member was expected to be present, and at which every subject which affected the interests or the work of the department was discussed. All changes in courses and all purchases of apparatus were discussed at these meetings.

In commenting on the general conditions existing at the beginning of the year I wish particularly to mention the cordial spirit with which all the members of the department received me, and the enthusiastic spirit of coöperation, which has made the work of the department very agreeable, and has made my work as a newcomer much easier than it would otherwise have been. Throughout the year, the staff has consisted of Professors Arendt and Morecroft; Mr. Hehre, Instructor; Mr. Mason, Assistant; Mr. Burnet, laboratory mechanic, and Prof. Slichter. With this staff of five teaching members, the department gave 22 lectures and 14 laboratory periods per week in the first term, and 26 lectures and 13 laboratory periods per week in the second term. Some of these periods included sections so large in number that I am doubtful whether all the students reaped the full benefit of the work. This applies particularly to the work of our direct-current laboratory which, in my opinion, has at present a capacity of 18 students for efficient work and in which it is positively inadvisable to instruct more than 21 students at the same time. The laboratory is in use six afternoons a week and two mornings. This means doing as much with the laboratory as is possible, with the present policy of morning lectures and afternoon laboratory periods. One man, Mr. Hehre, is in charge of the direct current laboratory and we have concluded that it was impossible for him to care for all this work, and in addition prepare experiments and correct reports, so that Mr. Mason and the writer arranged that one or the other was generally to be present to assist in the laboratory.

During the past year the electrical sections of the senior and junior classes have been rather small, and the greater portion of the work of the department has been with students of the other departments. Thus, this year there are only 10 graduates in electrical engineering. This

is a condition which we regret and hope that time and circumstances will change. It is some compensation to know that these men were of more than average ability, and a gratification that the graduating class has received more offers of positions than there were men.

With regard to the large size of some of the class sections, it is proposed as an experiment during the coming year to divide these sections into smaller groups, allotting their time between one of the senior members of the department and one of the instructors. That is, one of the senior members of the department will have immediate supervision of the course of instruction and will meet the section no less than once in two weeks and the instructor will take the other periods.

The Electrical Engineering Society, under the presidency of Mr. C. C. Hale, has held a number of well attended meetings. The subjects treated were quite varied and the speakers came from all parts of the country. Among our own graduates who addressed the society were Prof. F. B. Crocker and H. G. Chatsin, '97, E. E. H. L. Mellen, 1912, has been elected President for the coming year.

W. I. S.

DEPARTMENT OF ENGINEERING DRAFTING.

(*Extracts from annual report of head of department for year* 1910-11).

The work of the department for the past two years has been so well systematized that no important changes were deemed necessary during the current year. Mr. S. C. Miller, senior instructor, was promoted to the grade of assistant professor, and assigned a seat in the Faculty of Applied Science.

The attendance in the several courses during the year was as follows:

Course.	No. of Students.	Course.	No. of Students.
1 or 2	185	5 or 6	83
3 or 4	190	7 or 8	207

The entire work of conducting courses 5-6 and 7 or 8, attended by second year men, devolved on Messrs. Miller and Harrington, who, in addition to this work, also took part of the instruction in courses 3 or 4 of the first year.

Mr. Jesse Beans, Assistant, took entire charge of the clerical work of the department, and also rendered valuable service in assisting in the instruction in course 1-2. The successful operation of the department is largely due to the zeal and faithfulness of the officers above named.

R. E. M.

DEPARTMENT OF GEOLOGY.

(*Extracts from annual report of head of department for year* 1910-11).

The work in the required courses is now so well systematized, and is so largely determined by the current necessities, that no very important changes can be introduced until the six-year course goes into effect, should this great improvement be finally adopted.

In a general way, events of more than usual interest have happened in the department during the past year. On receiving sabbatical leave

for a half-year, Professor Grabau sailed for Europe in June, 1910, and utilized the seven or eight months until February, 1911, in visiting the most instructive exposures of the entire geological section of Great Britain and the Continent. Many collections of valuable fossils and specimens have come to us in consequence, and Professor Grabau himself has gathered a vast amount of experience which is now serving to improve our courses in stratigraphical geology. Both Professor Grabau and I were in attendance at the Eleventh International Geological Congress, in Stockholm, in August.

Prof. Berkey has brought out through the office of the state geologist a bulletin (No. 146) on the geology along the new Catskill aqueduct. The book contains 283 pages and 38 plates and is practically a monograph upon geology applied to engineering. Besides the problem presented by the great Ashoken dam, there are 15 or 20 miles of tunnels, whose geological sections had of necessity to be determined with great accuracy and detail as a basis for the letting of contracts. The careful geological determinations have prevented back claims and disputes almost entirely.

The undersigned sailed early in last July to attend the International Geological Congress in Stockholm, but previously to its assembling went on a three weeks' excursion to visit the Archean rocks and the iron mines of northern Sweden. The greatest single work of the congress was the preparation of a large two-volume book on the "Iron Resources of the World," in which I had collaborated. At the congress, a special morning was given to the discussion of the question, "Where shall the Iron Industry of the Future find its Supplies of Ore." The discussion was opened by the Prime Minister of Sweden, who was followed by Professor Hj. Sjögren of Stockholm, summarizing the large work just mentioned; by Prof. L. De Launay of the Higher School of Mines in Paris, for France; by Prof. Adan de Yarza, of the School of Mines in Madrid, for Spain; by Prof. H. Beyschlag, Director of the Prussian Geological Survey, for Germany; and by Prof. J. F. Kemp for the United States. The discussion was to have been closed by Prof. Henry Louis, of Newcastle, speaking for Great Britain, but Professor Louis was unavoidably absent. In December, at the annual meeting of the Geological Society of Stockholm, the undersigned was elected one of the 20 foreign corresponding members. In April came also an election to the National Academy of Sciences of our own country.

J. F. K.

DEPARTMENT OF MECHANICAL ENGINEERING.

(*Extracts from annual report of head of department for year* 1910-11).

The total amount of instruction given by the department was slightly in excess of that given last year.

It is with regret that we have found it necessary to give up three courses for next year: on manufacturing plant, organization and management, and refrigeration, all of which have been given by volunteer lecturers in the past and who are now unwilling to continue, some of them having served three years. We feel very grateful for the promise of $2000 to enable us to pay some of the other lecturers whose courses

otherwise would have to be dropped. This sum will be sufficient to cover the courses on shop processes, elevating and conveying machinery, air machinery and pumping machinery. We hope that this assistance, while it is inadequate to enable our work to go on as it should, is an indication of the proper provision to be made in the near future.

Excessive undergraduate demands on the instructing staff have made it necessary for us to withdraw from the graduate instruction in the University entirely, and as this is much to be regretted it must be regarded as a purely temporary arrangement to be removed as soon as possible.

Attention has been called in the past to the overloading of the instructors in this department, and to the excessive number of students in the various courses of instruction, with all the evils of unrest and inefficiency that result therefrom. There appears to be no relief in sight although it is just as much needed as when first pointed out. The arguments for it need not be repeated here as those already advanced still hold good. The same is true with regard to provision of room space, offices, library, study room, laboratory and shop, all of which are so urgently needed and to such an extent as cannot be satisfied except by additional building facilities. With regard to laboratory, not only is more space needed but additional equipment as well, whereas with regard to shops, all that can be said is that there are none worthy of the name.

STUDENT HOURS FOR ONE WEEK

Mechanical engineers: Lect.-recit.....1570; Lab.-draft.....	1876
Other students: Lect.-recit.....1864; Lab.-draft.....	1163
Total student hours for one week	6473
Total assigned student-hours for one term given to candidates for M.E. degree, lecture and recitation	21,980
Total assigned student-hours for one term given to candidates for M.E. degree, laboratory and drafting room	26,264
Total assigned student-hours for one term given to candidates for other degrees, lecture and recitation	26,096
Total assigned student-hours for one term given to candidates for other degrees, laboratory and drafting room	16,282
Total.....	90,622

DEPARTMENT OF METALLURGY.

(*Extracts from annual report of head of department for year* 1910-11).

Instruction.—But few changes were made in the courses given, and practically no change in the number of hours for the different courses; in all, a total of 23 courses were offered and students were registered in all of them.

The total number of students for one or more of these courses was 317, as follows:

Special students	3
School of Chemistry	29
School of Engineering	157
School of Mines	120
Candidates for Advanced Degrees	8
Total	317

A tabulated statement of the number of students registered in the principal courses is given below:

No. Course.	Subject.	No. Students.
101	Introduction to metallurgy	60
111	Metallurgy of copper	64
122	Metallurgy of lead, silver, gold and zinc	72
131	Metallurgy of iron (short course)	66
134	Metallurgy of iron (long course)	127
141	Electrometallurgy	71
156	Metallography (short course)	31
62S	Summer field work	57
72	Metallurgical laboratory	61
171	Metallurgical laboratory (long course)	7

The other courses given are more or less special and the number of students attending each are comparatively few.

Organization.—Within the past year, Dr. Kern has been promoted from the rank of instructor to that of assistant professor. The need, however, of the services of a regular assistant has been very seriously felt, especially in the work of Prof. Campbell and Dr. Kern in the laboratories. As conditions are at present, it is possible to do only a very limited amount of research work and the number of post-graduate students which can be registered for advanced degrees in the department is also limited.

Research.—The amount of research performed was not so great as during the previous year, which is to be regretted, but it was all that we could supervise. Several important pieces of research were completed, some of which were started the previous year, and some of which were started and completed during the past twelve months. Results of some of this research work will be published in the SCHOOL OF MINES QUARTERLY from time to time.

Summer Work.—The summer work in metallurgy consisted of visits to the various plants in the vicinity of New York, and also, for the metallurgical engineers, to plants in the vicinity of Bethlehem, Harrisburg, and Pittsburg. As this year there were not enough candidates to make up a party to go to the far West, we were again compelled to confine our work to the East. The number of metallurgical students coming up from the lower classes, however, should enable us to arrange for an extensive summer trip to the West in the summer of 1912.

New Assay Laboratory.—All the preliminary steps necessary to transfer the assay laboratory from the department of chemistry to the department of metallurgy were completed and the work of installing this laboratory on the sixth floor of the School of Mines building was finished during the summer of 1911. The laboratory is up-to-date in every respect, and is so arranged that the students can accomplish very much more work than was possible in the old location in the cellar of Havemeyer building. A. L. W.

DEPARTMENT OF MINERALOGY.

(*Extracts from annual report of head of department for year* 1910-11.)

The number of students taking courses under this department is the largest in its history. The increase was particularly in the courses taken by the civil engineers and chemical engineers.

Mr. Rogers, assistant in mineralogy, has been appointed Assistant Geologist on the Land and Classification Board, and will be succeeded by E. L. Bruce, a recent graduate of Queens College, Kingston, Ont., who, in addition to his regular course, has done good field and laboratory work in mineralogy under Prof. Nicoll.

Improved Equipment.—The large classes having quite outgrown the existing student collections, the following changes have been made:

(a) The student collection proper, as it has existed from the earliest days of the school, consisted of a fixed number of specimens in definite order, consecutively numbered and accompanied by correspondingly numbered cards on which the species represented were stated. It could only be extended by either destroying the order or leaving blanks to be filled in later, or, as was done some years ago, by building a second collection with its own set of numbers.

This year the plan was adopted of assigning the same number to each specimen of the same species (or each prominent variety), increasing the drawer space, and labeling only the drawer and merely with a key to the numbers included. By this scheme it is possible, whenever any specimen is obtained which it is desirable to add, simply to put it in with its appropriate number, and similarly any maltreated specimen can be taken out. It corresponds to a loose-leaf index. The number of specimens has been greatly increased.

(b) The duplicate set collections used at the lectures were increased from 20 to 35, involving the purchase of much new material and the construction of cases to hold the sets, for which a special appropriation was made.

(c) A number of new crystal models have been purchased, chiefly to replace models lost or destroyed.

Departmental Library.—The library has been materially increased by transfers of more of the older works from the general library and by purchases. It is hoped to include not only all modern works but all the older works which have played any part in the development of either crystallography or mineralogy.

Egleston Museum.—Comparatively few purchases have been made for the museum; these, however, include several unrepresented species as well as fine Brazilian phenacites, a number of better crystals of moderately rare minerals and a number of recent finds in new localities. In addition Mr. Dufourcq has presented some fine argentite crystals and Mr. Canfield some of the newly described thomsonites of Bergen hill, N. J.

A. J. M.

DEPARTMENT OF MINING.

(*Extracts from annual report of head of department for year* 1910-11).

In the sixteen courses given by the department of mining, the average number of students for the year 1910-11 was 51. It may be added that 46 students graduated last June.

Professor Munroe was absent on leave in Europe during the second half-year. He visited Sicily, Italy, Switzerland, Germany, Holland and England. Professor Peele spent a first-term leave of absence in making a visit to the gold mines of the Transvaal and the Kimberly diamond mines; incidentally going to Victoria Falls and the Zambesi, and later, via Cape Town and Madeira, to Argentina, Chile, Peru, and thence home by the Isthmus of Panama.

Professor James F. McClelland (S. of M., 1900), head of the department of mining at Yale University, gave several of the lecture courses in both terms during the leaves of absence above mentioned. His lectures were much appreciated.

The ore-dressing laboratory was operated as usual continuously throughout the second term, the 56 students who registered for it being divided into squads of from 10 to 13. Each squad had three weeks of practice in consecutive afternoons. The work was in charge of Mr. Judd, assisted by Prof. Kurtz when the large machines were in operation. It would be desirable to extend somewhat the time given to ore-dressing practice, but it is not at present possible to do so; the existing schedule for fourth-year students is already too crowded, especially in afternoon work. This condition is caused chiefly by the amount of time given to design of mine plant (four afternoons per week first term) and to the graduating thesis. Both of these subjects have proved to be so valuable that they should not be cut down in quantity beyond the present requirements. The time allotted to the design of mine plant is already less than formerly, caused by the necessity of finding room for practice in the alternating current laboratory. It is hoped that more time can be found for ore-dressing when the proposed six-year schedule goes into effect.

Departmental notes concerning the summer field work in mining and mine surveying are published each year in some detail in the November issue of the QUARTERLY and are therefore omitted here.

In the second term Mr. Judd made a short professional trip to Nevada to examine and report on a gold property. Last fall appeared a new edition of Peele's "Compressed Air Plant," which has been enlarged by nearly 200 pages of text and illustrations.

R. P.

BOOK REVIEWS.

Die Metallurgie des Wolframs, mit besonderer Berücksichtigung der Elektrometallurgie sowie der Verbindungen und Legierungen des Wolframs, samt seinen Verwendungen. By DR. HANS MENNICKE. 416 pp. Berlin, M. Krayn, 1911. Price 17 marks.

The complete title of this book is a good index of the scope its author intended it to cover. It deals not only in a very comprehensive way with the metallurgy, including the electro-metallurgy, of tungsten, but it also describes the preparation and characteristic properties of practically all better known tungsten compounds, including at least the simpler of the complex tungsten acids and their salts. The metallurgical chapters of the book are very complete. In fact, the author has been so anxious to provide against every possible presence of foreign metals, which may or may not be recovered, that it is not always easy to make out the best procedure in any one given case, a fair amount of cross-reference being necessary. Of course this condensed style, in which the whole book is written, enables the presentation of far more real facts than would otherwise be possible. In these chapters the author describes methods actually preferred and used by him for the production of tungstic acid and tungsten powder, and gives statements of cost of manufacture, which, however, are true probably only for Germany. It is a pity that no comparison with the cost of the direct processes of making tungsten steel and tungsten alloys from ores or impure tungsten compounds is made. According to the figures given, a tungsten works making up to 300 tons per year of tungsten yields a reasonable financial return, though of course there is danger of increased competition.

A good deal of information about the rarer metals found with tungsten is given. Scandium especially is fully dealt with, although nothing is said about any possible use. Weiss and Landecker's method for the separation of columbium and tantalum, described on p. 126, is but little known, and should certainly, if really reliable, displace all the older methods.

The chapters on tungsten wire for electric lighting, and tungsten alloys and bronzes, represent very full compilations of descriptions and patents of processes, a large number of references being also given; but here the author hardly ever commits himself to any critical expression of his own views or experience. The description of fire-proofing by tungsten compounds is disappointingly brief.

The book is well printed on good paper, but is somewhat marred by the occurrence of a rather large number of typographical errors, especially in the earlier chapters. A few of the more important are: *Tungstone* for *tungsten,* twice on p. 9; *Stobzit* for *Stolzit,* p. 18; *Moh* for *Mohr,* p. 20; *Roscoë* for *Roscoe,* p. 20 and elsewhere; WCl_{24} for W_4Cl_{24}, p. 29; $WOCL_4$ for $WOCl_4$, p. 39; *Kaliumsalzes* for *Kalzium-*

salzes, p. 134; *Gen. Elektric Co.* for *Gen. Electric Co.*, p. 200; *Firming* for *Firminy*, p. 308; *Harbort* for *Harkort*, p. 315. On p. 215, the statement that "[die Reduktion[war . . . . hinsichtlich WO_3 gelungen" should read "*nicht* gelungen;" the compounds referred to on p. 297 are not tungsten chromates but chromic tungstens; and the statement on p. 320 that under certain conditions the tensile strength of tungsten steel falls to "3000 ko." is unintelligible, unless perhaps kilograms per square centimeter are meant.

In spite of such minor blemishes, the book is indispensable to anyone interested in tungsten or its compounds. ARTHUR G. LEVY.

A Short Course in The Testing of Electrical Machinery for Non-electrical Enginering Students. By J. H. MORECROFT, E.E., B.S., Assistant Professor of Electrical Engineering, Columbia University, and F. W. HEHRE, E.E., Instructor in Electrical Engineering at Columbia University. New York, D. Van Nostrand Company, 1911. 154 pp. and 84 illustrations. $1.50.

The material contained in this book is a development of the work given in the electrical engineering laboratories to all the non-electrical students in the schools of applied science at Columbia University. The text is about equally divided into two sections, direct-current and alternating-current practice respectively. Under the first heading is given a list of 11 experiments and under the latter a list of 9 experiments, which constitute the essential tests of electrical machinery.

The subject matter is well prepared. It covers in a clear, concise and logical manner the fundamental principles, connections, test and normal operating characteristics of those forms of electrical machinery which the non-electrical engineer is most likely to encounter in practice. M. A.

Metallurgy, a Brief Outline of the Modern Processes for Extracting the More Important Metals. By W. BORCHERS. Authorized translation by WILLIAM T. HALL and CARLE R. HAYWARD. New York, John Wiley and Sons. 8vo, v + 271 pp., 218 figs. Cloth $3.00 net.

The preface states that the purpose of the book is to present, in concise form, the different processes used for extracting the important metals from their ores, and for refining them. Both the student and the practical engineer can thus get a broad view of modern metallurgical operations, without taking the time required for a study of the more detailed treatises on the subject.

The book is divided into sections, each dealing with a metal: Gold, Platinum, Silver, Mercury, Copper, Bismuth, Lead, Tin, Antimony, Nickel, Iron, Chromium, Tungsten, Cadmium, Zinc, Manganese, Aluminium. That a short text-book of metallurgy is needed must be acknowledged, and the author has performed the difficult task of condensation in an excellent manner. The present volume will prove of great assistance both to the student and to the practical engineer, but the reader without any previous knowledge of metallurgy will soon get into difficulties. The chief criticism against the book must be that it contains no introductory chapter on general metallurgy. A begin-

ner cannot be supposed to know the significance of a reverberatory or of a singulo-silicate. Other faults are small ones, such as the omission of any mention of the Parkes method of precipitation in the cyanide process (page [illegible]) or of the Lloyd-Dwight method of roasting (p. 126) and so forth. Almost six pages are devoted to the constitution of the iron-[illegible] series, whilst alloys of the other metals are dismissed in a few lines.

Taken as a whole the book will be acceptable to the practical engineer who knows some metallurgy or can be used as a text for a lecture course for students, the lectures supplying the fundamental ideas underlying general metallurgy. Binding, paper, printing and illustrations are all excellent. W. C

Mining Without Timbering. By ROBERT BRUCE BRINSMADE, B. S., E. M., New York, McGraw-Hill Co., 1911. Large Octavo, XIII + 309 pages, 146 figures, cloth, $3.00 net.

Professor Brinsmade is the author of a number of articles in various mining journals describing mining operations in different parts of this country. In this book he has reprinted many of these articles with numerous descriptions of mining methods by other authors. These contributions have been grouped in chapters according to the method of mining adopted. A large part of the book is devoted to the methods of mining recently developed in this country for the working of large orebodies.

In his preface, Professor Brinsmade states that the work is not intended for a complete treatise on mining and that emphasis is placed upon practical mining problems rather than upon engineering questions. He has also endeavored to supplement and not to duplicate material to be found in existing treatises on American practice and for this reason has omitted hydraulic mining and dredging, and has given minor attention to mine timbering, foreign methods of mining and the financial side of mining, which are well covered by Bowie, Storms, Ingalls, Hoover, Finlay, Mayer and others.

The greater part of the book is devoted to detailed description of methods of mining ore and coal and supporting the excavations at selected mines in this country and in the British colonies. These descriptions cover 60 examples which, as before stated, are grouped in chapters according to the method used. Many of these descriptions are excellent and contain the essential data as to the conditions under which the work is done and which have determined the final choice of the method adopted. In some cases, however, the descriptions are incomplete and unsatisfactory and the language used is not sufficiently clear or exact to make it possible to understand easily or fully the methods described.

The book bears evidence of hasty and careless preparation. The pages are marred by frequent typographical errors and in one place, on pages 98, 99, 100, a number of paragraphs belonging to one example have been transferred bodily to the description of another mine.

In several instances, as on page 160, the compositor, or his devil, has been permitted to use fiendish ingenuity in the arrangement of cuts on

the page, and in other cases, as on pages 109 and 110, the same unrestrained infernal influence has placed plan and section of the same workings back to back on opposite sides of the paper.

Professor Brinsmade has been forced to adopt a new method of classification in order to provide for the newer methods of mining, and while this classification is not altogether satisfactory and will doubtless be modified in future editions, nevertheless it serves a useful purpose in grouping similar methods of work together. Unfortunately Professor Brinsmade has not always adhered to his own system of classification and descriptions in several instances have been placed in the wrong chapters. It is to be hoped that a new edition of this work will shortly be needed and that the author will be enabled to remedy these very serious defects in an otherwise excellent and valuable work.

Professor Brinsmade has coined a new word "hangwall" which is a doubtful improvement on the old and characteristic term "hanging wall" in common use. The latter expression is doubtless derived from the idea of an overhanging wall which derivation is entirely lost in the new expression. Fortunately he does not always insist upon the new spelling, and it is hoped he will see the error of his ways in the future editions of the book.

H. S. M.

PERSONAL ITEMS

S. F. Shaw, E. M., 1903, consulting engineer, has come back from Los Angeles and has opened an office at 136 Liberty St., New York City.

Robert K. Painter, E. M., 1896, has been spending some time with business associates in the Sturgeon Lake District, Ontario, where he has extensive mining interests.

J. C. Mabden, Jr., E. E., 1897, has been elected one of the vice-presidents of the Sloss-Sheffield Steel and Iron Co., Birmingham, Ala., and will act as managing director of the concern.

The class of 1911 is pretty widely scattered, according to information furnished by the Secretary, C. G. Mitchell, Britannia Beach, B. C., and dated September, 1911. P. R. Iseman is assistant foreman of one of the Yukon Gold Company's dredges at Dawson, Y. T.; L. S. Thompson is working (P. O. box 42) at Cripple Creek, Colo.; P. D. Wilson is assayer at the Copper Queen Company's smelter at Douglas, Ariz.; C. E. Dwyer is working for the Federal Mining and Smelting Company, at Mullan, Idaho; D. B. Scott, H. Banks and R. Franke are on the staff of the Miami Copper Company, Miami, Ariz.; C. G. Mitchell, W. L. Thomas and T. M. Bains are with the Britannia Mining and Smelting Company, Britannia Beach, B. C.; F. A. Ayer is assistant geologist with the Detroit Copper Company, Morenci, Ariz.; R. L. Strobel is at the A. S. & R. Company's plant at Murray, Utah; H. P. Corsa and J. C. Guiteras are with the Bingham-New Haven, and F. A. Wardlaw is with the Highland Boy, at Bingham Canyon, Utah; A. Mendlesohn is with the Franklin mine at Houghton, Mich.; J. M. Hoffman is prospecting at Porcupine, Ont.; M. L. Lee is agent for Clark, Dodge & Co., at Basin, Mont.; C. Pigott is chemist with the A. S. & R. plant at East Helena, Mont.; and W. R. Wright has recently gone to the Braden Mine, Rancagua, Chile.

Edward F. Pelton, E. M., 1903, is geologist for the Detroit Copper Company, Morenci, Ariz.

Recent alumni visitors at the School of Mines: A. W. Geiger, E. M., 1905; C. C. Swartz, E. M., 1904; P. L. Roberts, E. M., 1908; J. E. Rypinski, 1910; J. C. Porter, E. M., 1905; C. L. Bruns, E. M., 1909; D. L. Hover, 1910; A. A. Heimrod, 1910; H. K. Masters, E. M., 1894; C. W. MacDougall, E. M., 1903.

William Newell, M. E., 1907, has been appointed mechanical engineer to the bureau of factory inspection in the Department of Labor of the State of New York. Soon after graduation he was employed by the

Fidelity and Casualty Company, of New York, to study the causes and the prevention of industrial accidents, and the pamphlet compiled by him in collaboration with an associate in the same concern has been widely issued and has received exceedingly favorable comment. His present position was created by recent act of the Legislature, based on views presented to the legislative committees by the Fidelity and Casualty Company. Mr. Newell was married on April 16, 1910, to Mary Elizabeth MacCracken, of Newark, N. J., and now resides in New York City.

P. D. Wilson, 1911, with the Copper Queen smelter at Douglas, Ariz., is the author of a valuable contribution on the use of niter in assaying, published in the *Engineering and Mining Journal,* Dec. 30, 1911.

Walter L. Hoyt, C. E., 1876, died suddenly at Argo, Ill., on Dec. 14, 1911, at the age of 50 years. Among his early professional engagements was the erection and operation of the Grant smelter, at Denver. He next went to Sydney, Australia, where he had charge of a smelter for four years. He then intended to retire, but was persuaded to take charge of a refinery at Perth Amboy, N. J. His visit to Argo, Ill., was for the purpose of selecting a site and superintending the erection of a smelter for the International Smelting and Refining Company, after which he had expected to return to Perth Amboy.

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THE

SCHOOL OF MINES QUARTERLY.

A JOURNAL OF APPLIED SCIENCE.

CONTENTS

PAGE

PATENTS. By *William H. Davis* 209

DIAMOND DRILLING AT PORCUPINE, ONTARIO. By *Albert E. Hall* 229

STUDY OF MECHANICAL CONVEYORS—II. By *Lincoln DeG. Moss* 233

MODERN INCANDESCENT ELECTRIC LAMPS. By *Leonard J. Lewinson* 259

PUMPING BY COMPRESSED AIR. By *Herbert T. Abrams* 293

BOOK REVIEWS 312

PERSONAL ITEMS 315

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THE
SCHOOL OF MINES
QUARTERLY

VOL. XXXIII. APRIL, 1912. No. 3.

PATENTS.*

BY WILLIAM H. DAVIS.†

Our patent law is based on that section of the Constitution which provides that "Congress shall have power to promote the progress of science and useful arts by securing for limited times, to authors and inventors, the exclusive right to their respective writings and discoveries"; but this section is an expression of a policy which had existed under the common law of England for centuries before the adoption of our Constitution. At least as early as the times of William Rufus, we find instances of the grant of open letters — letters patent —securing to merchants or guilds the exclusive right to exercise specified trades or manufactures within the English realm, these grants being for the purpose of encouraging the introduction of trades, and the development of commerce in articles which were not produced within the realm. Since these grants were quite profitable, they were commonly purchased from the Sovereign by the payment of a lump sum, and it came to pass that in times of financial stress, from which even Sovereigns are not exempt, such exclusive privileges were granted with respect to articles of commerce which were previously known and freely bought and sold within the kingdom. Particularly in the reign of Elizabeth, these grants were made for practically all of the necessities of life, even for such an article as salt, and under them the people were horribly oppressed. The English courts,

* Lecture before the Electrical Engineering Society of Columbia University, 1911.

† Of the New York Bar.

which were always faithful to the people, consistently held that all such grants were unlawful unless they related to the introduction of a new manufacture within the realm; but since they could not directly attack the validity of a royal grant, they had to content themselves with refusing to aid in its enforcement.

In the times of Elizabeth, the Parliament so clearly indicated its purpose to prohibit such grants in future that Elizabeth promised to discontinue the issuance of them, but this promise was not faithfully adhered to by her or by her successor, James I, so that in 1623, in the reign of James I, Parliament enacted the Statute of Monopolies, in which it referred to the burdensome conditions of such grants, denied the right of the Sovereign to issue them, and made them forever unlawful, with the single exception of grants for limited terms with respect to manufactures newly introduced into the realm, and then only if they were not mischievous to the State by raising the price of commodities at home, or hurtful of trade, or generally inconvenient. This statute is the present basis of the patent law of England.

Exercising the power conferred upon it by the Constitutional provision to which I have referred, our Congress has enacted the patent statute and the copyright law. I wish to confine my remarks substantially to the patent law, the law under which are issued what we call "mechanical patents," though I prefer the expression used abroad, "patents for the protection of industrial property." Nevertheless, it may be well to distinguish briefly between "mechanical patents," "copyrights," "designs" and "trade-marks."

Under the patent statute, patents are granted for new and useful arts, machines, manufactures and compositions of matter. These things are characterized by utility in the practical sense. The same law also provides for patents for designs, which must be ornamental as distinguished from useful in an industrial way. The copyright law gives protection for the writings of an author, including dramatic and musical compositions, maps, works of art, drawings or plastic works of a scientific or technical character, photographs, prints and pictorial illustrations. Copyrights are registered in the Library of Congress. If a thing is a work of art to be used in connection with articles of merchandise, such, for example, as the "Campbell Kids," it must be registered in the pat-

ent office, but under the provisions of the copyright law, as a print or label. If the thing is an arbitrary sign, word or mark, intended to indicate, by association, the origin of goods in trade, it is a trade-mark, and is registrable in the patent office under the provisions of a Federal trade-mark statute. The mark must not be descriptive of the goods nor of the manufacture but must be arbitrary, and the more arbitrary the better.

What May Be Patented.

Our statute says that a patent will be granted to anyone who has invented or discovered any new and useful art, machine, manufacture or composition of matter. The word "art" includes processes, methods of procedure in themselves intangible and known only by their industrial results. "Machines" and "manufactures" cover a very wide field, including all visible and tangible machines and organizations of machinery, as well as instruments and tools. The words "composition of matter" define themselves, and permit an inventor to secure in this country patents for new chemical compounds or mixtures of substances, which is not commonly permitted in the European countries.

The thing must be useful in the sense that it is operative and not harmful or contrary to public policy, but beyond that the law does not inquire into its degree of utility. If the thing will work, that is enough to satisfy the statute, whether it is more useful than previously existing things or not.

The statute says that the thing must be new, and it goes on to define what novelty is by saying that it must not have been known or used by others in this country before the invention or discovery of it by the person seeking the patent; and not patented or described in any printed publication in this or in any foreign country before his invention or discovery thereof or more than two years prior to his application for United States letters patent. It should be noted that the knowledge or use of a thing in a foreign country will not bar a patent in this country unless the thing has been patented or described in a printed publication abroad. That is evidently just, because the object of the law is to promote the progress of science and useful arts *in this country,* and the thing might be used abroad without anyone ever knowing it

here. But if it is patented or published abroad in a printed publication, such as a technical journal, it is pretty safe to assume, in these days, that it will be known here. At any rate, it could be known here if our citizens were diligent enough to look for it. The prior publication must, however, be so complete as to enable anyone skilled in the art who reads it to practise the invention without experimentation involving more than the expected skill of persons in that art.

So, too, the law guards against the inventor's keeping his invention to himself too long, by providing that if the publication in this country or abroad is more than two years prior to his application for patent, it will be a bar even though he can prove that he invented it before the publication. So, too, if the thing has been in public use or on sale in this country for more than two years prior to the application, no patent will be granted.

The statute also says that the thing must be invented or discovered, and this is the great battlefield of the patent law. It has been said that it is impossible to define "invention." We find in the books a great deal of speculation about the character of the mental act involved, but this has led us only to the conclusion that the character of the mental act is unimportant. The invention may result from years of laborious thought, including the study of the ideas of others, and long experimentation. An example of such a case is the Edison incandescent lamp, the carbon filament of which was arrived at by a long process of elimination, which is characteristic of Mr. Edison's methods of procedure. Or an invention may result from a purely accidental circumstance, impressing itself upon a ready and observant mind. An old example of this is the invention of the water-tabby. The inventor spat upon his table and then laid his hot iron on it and observed that a mark having a flowery appearance was produced. He then tried it on linen and found that it produced the same effect. It is said that the linen so marked was popular for many years, and that the inventor made a large fortune. The fact is that it is not humanly possible to know the character of the mental operations which result in an invention, and the law should not, and it does not, consider this, but looks rather at the result produced, whether it constitutes progress in science or the useful arts.

I will not attempt to define invention where every one agrees that it is not capable of definition, but I would like to suggest a practical test. If you are conscious of an existing want—of an impediment in the art in which you are trained, or in any other art—and by your efforts remove that impediment and satisfy the want, you ought to have a patent unless you find that someone is ahead of you and has satisfied the same want in the same way. And when I make this exception, I ask you not to let the patent office or your attorney find part of your invention in one place and part in another place and then put the two together and say that the putting of them together was obvious. Your reply is that wants are not felt if there is an obvious way of satisfying them. The fact is that as the arts progress, the mass of previously existing relevant material accumulates, and the chance correspondingly increases that after the invention someone will be able to find in the prior art things which, after the event, appear to be suggestive of it. But at the same time the dissemination of knowledge, and the number and activity of the minds engaged in developing the art, likewise increase, so that the prior existence of the want, without satisfaction, becomes increasingly sound evidence that the production of the invention was not an obvious exercise of the expected knowledge and skill of such minds. Do not forget that your invention must be an art, machine, manufacture or composition of matter. You cannot have a patent for the discovery of a law of nature, but only for the application of that discovery to a useful purpose. So, also, you cannot have a patent for applying your technical skill in laying out a generating station, for example, though you may give a great deal of careful study to the selection of the proper type and capacity of the units, to the arrangement of the circuits, and so on; nor for a scheme of financing; nor for a system of shop practice suitable to develop to the highest degree the efficiency of the men and the machines employed; because these things are not arts, machines, manufactures or compositions of matter. Even in connection with arts and machines you cannot have and ought not to expect a patent for the application, to the particular conditions involved, of that engineering skill which is common to men of your profession. If patents were granted for that, they would

tie your hands and those of your fellows, and would not aid but would deter the progress of science and the useful arts.

What a Patent is.

Let us suppose that you have made an invention which is the proper subject matter of a patent, and let us see in what way the patent system protects you. Imagine that you are the inventor of the electric motor. Prior to your invention the thing did not exist and, of course, nobody could use it. After your invention it does exist and (assuming that no one has a monopoly of the parts of which it is made or of means for developing the current for driving it) you can use it. Without the patent law you would have no way of preventing others from using it unless you keep it secret and, of course, in the case of the electric motor that would be impossible because any extended use of it would necessarily disclose its nature. Now in this situation, the government steps in and offers to make a bargain with you. It says, if you will make a full disclosure of the invention so that anyone skilled in the art can practise it, and will file that description in the patent office, and take certain other formal steps, including the payment of $35 to the government, the government will grant you the *exclusive* right to manufacture, use and sell the invention within the territory of the United States for seventeen years from the date of the patent. Please note that the emphasis is on the word "exclusive." The right you get is the right to exclude others. In the supposed case, you already had the right to use it, but if you had not had it, the government would not pretend to give it to you. It is very important to understand that aspect of the matter. Suppose that after you had obtained your patent for the electric motor, someone came along and invented the carbon brush as a substitute for the copper brush which you had been using. Before this new invention was made you could not use the carbon brush because you did not know anything about it. The man who invented the carbon brush is entitled to a patent for it and to exclude others from using it. But he has no right to use it himself because you have a patent covering the electric motor, and he cannot use his brush without infringing your patent on the motor. For some reason that situa-

tion does not seem to be easily comprehended, but a little consideration will show that it is the only possible condition. Prior to the invention of the carbon brush, the inventor of it had no right to use your motor and there is nothing in the nature of that invention which would give him such right. If there were, your patent would be worthless because every one who invented an improvement in electric motors could ignore it. In other words, a patent gives a right to exclude others from manufacturing the invention and does not in any case purport to grant the right to manufacture the patented thing. If such manufacture is an infringement of a prior existing patent, it is just as much an infringement after the grant of the patent for the improvement as before.

Procedure to Obtain A Patent.

We may now ask how you are to go about getting a patent after you have made an invention. The first thing you should do is to seek the aid of an attorney, whom you should treat with the utmost confidence, disclosing to him all you know about your invention and everything that may seem to you to have any bearing upon it. Everything depends upon the character of the services rendered by your attorney, and since he must necessarily be dealing every day with many different arts you cannot expect him to be an expert in the particular art to which your invention relates. Moreover, I do not hesitate to say that in all probability you will not yourself know exactly what your invention is in its real essential character, and in its relation to the prior art. Your attorney has to find that out and more than that he has, so far as he can, to look ahead and understand what relation your invention is going to bear to the future development of the art. I have heard it said that a patent solicitor ought to be a lawyer, an engineer, an inventor and a prophet—and so he ought, but he seldom is. Consequently, you ought to put into his hands every detail of your invention as you know it, and assist him to the fullest extent with your technical knowledge.

After you have disclosed your invention to your attorney, he will prepare papers for an application. Among these will be a specification and drawings. The law requires that the drawings shall illustrate and the specification shall describe the best

form of the invention known to you on the date of the application. The drawings have to be made on sheets of a certain size in conformity with very stringent patent office rules, and are commonly made by persons skilled in that particular line of work. The specification, so far as it constitutes merely a description of the invention, must be so full, clear, concise and exact as to enable anyone skilled in the art to practise the invention. An accurate description of the invention, considered merely as such, could doubtless be drawn by any careful, educated person acquainted with the art to which the invention relates. You will find, however, at the end of the specification of every United States patent one or more numbered paragraphs, constituting concise and definite statements of the invention, and preceded by the words: "What I claim is," or words of like import. Everything depends upon these claims. They are the soul of the patent, and as such, should be reflected in the body of the patent, that is, the descriptive portion of the specification preceding them. It is this that makes the descriptive portion hard to draw. A properly drawn patent is a work of art of which every part supports every other and all aim at the same end, a statement of the real essence of the invention, which is epitomized in the claims.

Perhaps I can illustrate how important these claims are by referring again to the supposed invention of the electric motor. Suppose your first conception of the motor corresponded to one of the forms which did historically precede the present type of motor, i. e., the form in which the armature was made up of a number of radially disposed projecting pole-pieces wound with the exciting coils. Now we may imagine that your attorney in claiming your invention, and not being a good attorney, made the following claim:

"What I claim is:

1. An electrical machine adapted to transform electrical energy into mechanical energy comprising a field magnet having pole-pieces, an exciting winding on the pole-pieces, an armature having a series of radially directed pole-pieces of a greater number than the pole-pieces of the field magnet, and exciting windings on the armature pole-pieces."

Such a claim as that could not be successfully used to prevent the manufacture of the modern type of electric motor having a

ring or drum armature. You would be bound by the limitation which your attorney had inserted in your claim, i. e., your patent would not cover any motor that did not include an armature having a series of radially directed pole-pieces of greater number than the number of pole-pieces of the field magnet. The idea will doubtless suggest itself to you that it is a hardship that the inventor should suffer from such lack of foresight on his part and the part of his attorney. And so it is. But the law proceeds on the theory, which I believe to be wholly sound, that it is essential to the progress of the arts that the general public should be able to ascertain with a reasonable degree of certainty what it can do and what it cannot do without infringing existing patents, and for this reason it is necessary, as a practical matter, to confine the inventor's monopoly to the subject matter expressly defined in his claims. If you ever become involved in a patent controversy in which such a question as this arises, you will find that the courts have relieved against the apparent hardship to a great extent by the so-called "doctrine of equivalents" which is to the effect that if the claim specifies a combination made up of a number of units, as all claims do, then the substitution by a competitor, for one unit of the combination, of a known equivalent thereof does not avoid the patent. For instance, if the claim specifies a spring as one element of a mechanical combination and the competitor uses a weight in a place where a weight is the known equivalent of a spring, infringement of the claim is not avoided.

A skillful patent attorney comes to adopt many different ways of trying to define the invention in claims of adequate scope. He will usually draw a claim as broad and comprehensive as he can and then put himself in the place of one desiring to avoid the patent and try to devise combinations of parts which will attain the desired end but which will not fall within the terms of his tentative claim. In this way he usually sees how the claim can be improved upon; and he usually repeats some such procedure with a number of claims which are differently stated because they view the invention from different standpoints. It would probably not be particularly difficult to arrive at a sufficiently broad statement of the invention, but the difficulty is that it should not

be too broad. The broader it is the more liable it is be overthrown by some patent or publication subsequently discovered in the prior art which, though it may have the underlying idea of the invention in its broadest aspect, may lack some characteristic which is practically essential to success. The result of these conditions is that the patents usually contain a series of claims of different scope, because if the broad ones are subsequently found to be too broad, you can fall back on the narrow ones. I am afraid, however, that under the pressure of the great responsibility, and in the presence of technical difficulties, the tendency for us is to indulge in too large a number of claims, hoping thereby to let nothing escape, rather than to prosecute our analysis until we get to the real essence of the invention.

After it is prepared and sworn to, the application is filed in the patent office, with a government filing fee of $15. In the patent office is a large force of examiners, something like three hundred, who examined and passed upon something like 65,000 applications for patents in the year 1910. They are mostly young men, graduates of our technical schools, who are selected by a searching competitive examination on technical and scientific subjects, and who receive salaries of $1,500 a year at the start. There are 43 examining divisions, each of which considers and passes upon a specified class or classes of applications Each application is sent, when it is received by the patent office, to the proper examining division and is there taken up in its turn by the examiner who has that particular line of work. Every week, in the *Official Gazette* of the patent office is published a report from each of the examining divisions showing the oldest unexamined application. On November 4, 1911, the division which had its work best in hand reported the oldest unexamined application as dated October 5, and the division which was most behind in its work reported the oldest unexamined application as dated July 24. The examiner carefully reads the specification and notes any typographical errors, or obscurities or inaccuracies of description, and then takes up each of the claims separately and makes an examination to ascertain whether the subject matter of that claim is old. He is assisted in this by having closely at hand a thoroughly classified set of the United States and foreign patents relating to his

particular subject, and by the fact that in the course of his work he comes to be pretty familiar with the art in which he is working. If he finds that any of the claims are old, or if he thinks they are not expressed in accordance with certain formal rules which have grown up, he makes note thereof. He then writes to the attorney stating his objections fully and making reference to all the supposed anticipations. The attorney then gets copies of these patents or publications, studies them in connection with the claims and modifies the claims if necessary, to distinguish them from the prior art. If he does not agree with the examiner that such modification is necessary, he presents an appropriate argument. The prosecution of the application proceeds in this way until the attorney and the examiner reach an agreement, or a final disagreement. If they agree, the application is allowed and upon payment of a final government fee of $20 the patent issues. If they disagree, an appeal lies to the board of Examiners-in-Chief, made up of three examiners of long experience. From their decision an appeal lies to the Commissioner of Patents, and from his decision, to the Court of Appeals of the District of Columbia. If the applicant and his attorney are not satisfied with this series of appeals they may file an appropriate bill in equity in a Federal court, and if the court agrees with them they may compel the Commissioner to issue the patent in the form requested. This provision of the law, however, is very seldom resorted to. Any modifications made in the claims during the prosecution should be made very carefully, for anything which the inventor deliberately gives up during such prosecution he can never again claim as part of his invention.

It happens in an extraordinarily large number of cases that more than one application claiming substantially the same invention will be pending in the patent office at the same time, these applications having been filed by different and independent inventors. In such cases, a proceeding known as an "interference" is instituted. The examiner selects from the two or more interfering applications those claims which he thinks are most appropriate to cover the subject matter common to the applications, and he suggests those claims to the other parties. The claims are then inserted in the respective applications and the whole matter is referred to the ex-

aminer of interferences with the purpose of commencing proceedings to decide who was really the first inventor. For this purpose testimony is taken in the manner which I shall describe in connection with suits to enforce a granted patent, with the exception that the testimony of experts is very seldom resorted to because the patent office examiners are themselves experts. After the testimony is all taken, the case is argued before the examiner of interferences and is decided by him. From his decision, appeals lie, as in the case of individual applications, to the board of Examiners-in-Chief, the Commissioner, and the Court of Appeals of the District of Columbia. In this case also, resort may be had, if the party is not satisfied, to the United States courts.

How Enforced.

We may now ask how a patent, after it is granted, is to be enforced. I will treat this only briefly, because if you ever get that far into the meshes of the law you will need much more and better advice than I can give in this limited space. You could bring an action before a jury for damages, just as for the taking of any other property unlawfully, but by bringing a special action in equity you can, if you succeed, recover the defendant's profits, which will often be more than your loss, particularly if you are a struggling inventor and not a large manufacturer. Furthermore, in an equity case you can get the really important relief in connection with a patent, an injunction restraining the other fellow from continuing his infringement. The procedure is to file a complaint charging the defendant with infringement of the patent, and he replies in an answer denying infringement if he can, and attacking the validity of the patent. That is, the defendant may say that the patent was mistakenly granted because the supposed invention was old in that there existed prior knowledge or use of it in this country, or prior patenting or publication in this country or abroad, of which the patent office did not know when it granted the patent, or if it knew, did not appreciate. Or the defendant may say that although the very thing disclosed in your patent was never known before in that exact form, yet it was so like the things that had gone before that anyone skilled in the art would produce it as soon as any need for it was felt.

There are other defenses in a patent suit, but those just referred to are the chief ones. After the answer has been filed by the defendant, the parties join issue and proceed to take testimony to prove their allegations. The manner of taking this testimony is extremely informal. The attorney representing the patentee gets together his witnesses and notifies the other side that he will take testimony on a certain day, and at a certain place, which is usually in the office of the attorney. On that day the attorneys and witnesses attend, the witnesses are sworn by a notary public, and their testimony is taken by questions and answers which are usually taken down directly on a typewriter. The attorney for the patentee offers the patent in evidence, and then calls a witness who describes the supposed infringement. This testimony is usually supplemented by the testimony of a man of scientific education, who is put forward as a person fully informed in respect to the art to which the invention relates, and who compares the patent with the supposed infringement and points out wherein the infringement exists; because it is correctly assumed that the Court will not, in most cases, have technical knowledge sufficient to understand the invention without such assistance. This usually takes several days, and often weeks. Then, at a later time, after due preparation, the defendant produces his witnesses, including perhaps persons who have knowledge of alleged uses of the thing in dispute before the date of the supposed invention; and finally a second expert witness who refers to and describes the patents and publications of the prior art which are thought to be pertinent. and points out wherein they disclose ideas corresponding to the alleged invention of the patent. Then after a further period of preparation, the complainant, the owner of the patent and instigator of the suit, calls further witnesses in reply to or rebuttal of the facts shown by the defendant's witnesses, and finally closes with his expert witness who attempts to refute the statements advanced by defendant's expert. All this occupies many more days or weeks.

The position of the expert witness in such cases is extremely important, and any one of you may sometime be called upon to occupy it. It is, I think, in connection with this expert testimony that the greatest abuses of the patent system exist. The manner of taking testimony above outlined does not provide for any super-

vision by a judicial officer, and there is no way to limit the scope of the direct and cross examination. The experts are permitted to express, in effect if not in terms, their opinion as to whether or not the patent should be sustained, and their testimony is, in fact, an argument in support of the patent or against it, as the case may be; whereas it should be strictly confined to an elucidation of the technical points involved. This unsatisfactory system of taking testimony is not limited to patent cases but is common to all equity suits in the Federal courts. A commission has been appointed by the Supreme Court to revise the rules of practice, and in my opinion the revision will result in the taking of testimony in the presence of a judicial officer, who will have the power to direct and confine it. If that is the result, it will come to pass, I think, that the men employed as experts in such cases will be men who are really skilled in the particular arts, rather than men who, as is too frequently the case at present, have only such knowledge of the particular arts as may be acquired from books, and are more expert in spinning beautiful but unsubstantial tissues which becloud instead of elucidate the facts, than in the real and practical aspects of the art with which they profess to deal. The testimony taken in this way is printed, and the case set for argument before a judge of the local District court on a certain day. At that time a printed copy of the testimony is handed to the judge, together with printed briefs for both sides, and the oral argument proceeds at once. The judge has never before heard of the case and the attempt in the oral argument is to lay the whole mass of technical facts, with the relevant points of law, before him in the time allowed, which may be from an hour or two to a full day, or more in extreme cases. In due time, the judge hands down his decision, with a written opinion. From this decision appeal lies to the Circuit Court of Appeals.

Assignment of Patent Rights.

I come now to a consideration of the ownership of the patent. In the first place, our patent law differs from that of every other country except Canada in the circumstance that the patent must be applied for by the inventor, and cannot be applied for by his agent or assignee. If it can be shown that the man in whose name

the application was made is not really the inventor, the patent is void. If the application is properly made by the true inventor, it can be assigned absolutely, before or after the patent issues, but I want to warn you on no account to assign to anyone any fractional part of the invention unless you couple with that assignment an agreement that no further rights shall be granted without the written assent of all the owners. A part owner of a patent, however small may be the fraction that he owns, has the unrestricted right to use the invention without accounting to the coowners, and since anyone who owns a part of the invention may divide that part into any number of smaller parts, and assign each of them to someone else, he has, as a practical matter, the power to destroy the monopoly. The same condition exists in the case of joint inventors, and in such case, also, an agreement should be entered into restricting the right of the individual joint inventors to assign or license without the written consent of the other joint inventor or inventors.

Rights of Employer and Employee.

A practical question which often arises is whether an employer has any right, and if so what right, to the inventions of his employees. If the invention is made outside of the scope of the employment, the employer has no right to it; but if it relates to the subject matter of the employment, as it usually will, and if it is made in the regular course of the employment, and the employer's materials and labor are used to reduce it to practice, or to experiment with it, the title to the invention and to the patent still resides in the inventor, but the employer will have a right to use the invention in his own business, which will usually be sufficient to limit greatly the value of the patent rights to the employee. The employee still has, however, the right to license others to use the invention, and that consideration is always a potent factor in inducing the employer to make fair terms with the employee. But I think there is a more potent factor in that regard in the broad consideration that it is always good business policy for an employer to treat his employees fairly, and I believe that as a general rule that policy prevails. It is, however, quite common for the big manufacturing concerns to make their employees agree to

assign to the company all inventions made in the line of the company's business during the employment. That policy has been criticised, and will perhaps appear to you unfair, but in my opinion it is eminently fair, because it is the only possible basis on which the big companies can do business. The large industrial concerns employ a multitude of men who are paid to devote their energies to the good of the company. Some of them are engineers and potential inventors, others are employed in the sales department, others in the planning and setting up of installations, and so on. It happens that the good ideas of the inventors can be protected by patents, whereas, generally speaking, the good ideas of the salesmen and the men who make and plan the installations can not. Every man who advances the interest of the company ought to receive a reward, but there is no reason why the inventors should be put in such a position that they can hold up the company and fix their own remuneration. Furthermore most inventions are not the result of the thought of one man but are rather the result of the work of many minds, to which the inventor has given only the final touch; and this is particularly true in the work of large industrial corporations in which each of the engineers is necessarily put in touch with the very latest developments in the art, and the problems most urgently needing solution, and is surrounded with an atmosphere, emanating from the combined efforts of the company's engineers, in which he is in a peculiarly advantageous position to make inventions, and into which he could not penetrate but for the circumstance of his employment. This is, to my mind, really the controlling consideration, since the companies would not dare to put such information in the hands of their engineers if they were not protected by an agreement that the inventions resulting should belong to them.

Foreign Patents.

Patent laws now exist in practically all enlightened countries, and they are all characterized by the fact that they grant for limited terms, the exclusive right to make, use and sell the invention. In almost every other respect, the foreign laws differ radically from our law. In most countries the application for patent may be made by anyone and need not be made by the inventor, as

in this country. The interference proceedings above referred to do not exist because the first applicant is conclusively presumed to be the first inventor, so far as the examination of the case in the patent office is concerned. In the Latin countries no examination as to novelty is made, and the patents are issued as soon as the applications are found to be in proper form. The government does not undertake to say whether the patented invention is new or not, but leaves that to be subsequently determined by suits in the courts. In Germany and most of the northern countries of Europe and in England, Canada and Australia, such examination is made much as in this country. If the application is found to be allowable in these countries, it is opened to public inspection and anyone who conceives that he would be injured by the grant of the patent can oppose the grant on the ground that the invention is not new, etc., etc. In this country, after a patent is issued, there are no further fees to be paid and no way in which the patent monopoly can cease before the end of the 17-year term. except by a court decision that the patent is invalid. In almost all foreign countries, on the contrary, the patent is looked upon as a source of internal revenue, and taxes must be paid at stated intervals, usually yearly. These taxes are commonly small for the early years of the life of the patent and increase progressively toward the end of the term. The idea is that if the patent is any good, it can stand the added expense, and if not, it ought to be permitted to lapse. In most countries, there is some provision to the effect that the patent will lapse if the invention is not "worked," i. e., manufactured or practised, to such an extent as to satisfy the demands of the public. In other countries the failure to "work" does not terminate the life of the patent but makes it possible for anyone who desires to do so to obtain a license on reasonable terms, which will be fixed by an appropriate tribunal.

I may further make reference to the International Convention for the Protection of Industrial Property. This is a treaty entered into by the United States and practically all of the important foreign countries with the exception of Russia and Canada. By its provisions, a period of priority of 12 months is secured to a person who files an application in any one of the contracting countries. That is to say, if you file a United States application

and within 12 months from the American filing date, file a corresponding application in any one of the foreign countries adhering to the convention, that foreign application will be treated as though it were filed abroad on the date that your United States application was filed. Of course, this is immensely important in foreign countries where the date of application is taken as the date of the invention.

Value of the Patent System.

To what extent, then, is the patent system of benefit to you? As individual engineers it enables you to prevent others from using your original ideas, and if you have no means of your own to develop these ideas it puts you in a position to bargain with others who have. This position is weakened by the great cost of prosecuting a patent suit. I hope that it will not be long before this cost is materially reduced, but I fear that it will never be reduced to such an extent that the average engineer, without independent means, can afford to defend his patent in the courts. Nevertheless, the position of the patentee is still pretty strong. It is a matter of good business policy on the part of the large corporation to encourage inventors to bring to them their patented or patentable ideas, and it is bad business policy for them to enter upon infringement of a patent even if the owner of the patent is a poor man. The larger the corporation the greater, in the nature of things, will be the extent of the manufacture and sales undertaken in violation of the patent, and though the corporation may know that the owner of the patent cannot afford to defend himself, it can never know at what moment the patent may be purchased by its largest rival and become effective to create for the infringer a disastrous commercial situation. On the other hand, if the patent is good, and can be sustained, the big corporation wants it as the basis for a profitable business enterprise.

As members of the general public you are benefited by the patent system just in so far as that system fosters and promotes the progress of science and useful arts—and that is very far. Its effect to direct and constantly impel thousands of ingenious minds toward a purpose to invent is unquestionably enormous, as is also its effect to encourage and sustain the individual inventor. In rendering unnecessary and unwise attempts to keep secret the mass

of industrial processes and scientific discoveries, it renders possible that free exchange of information and ideas which characterizes our technical journals and the meetings and publications of scientific societies. And in launching large enterprises based on inventions or discoveries it renders indispensable service in the encouragement of capital. I have heard it said by a man who knows, that in the early days of the Bell Telephone Company the large and necessary expenditures would not have been possible but for a reasonable assurance of patent protection. It is also unquestionably true that the important research laboratories which are maintained by such concerns as the General Electric Company depend for their existence on the patent system.

Like every power for good, the patent system may undoubtedly become, in unscrupulous hands, a power for evil if not properly controlled, but in my opinion, the Sherman or Anti-Trust Act, in the hands of the Federal courts, is entirely adequate for such control. It seems to me that the principles which are to guide the application of this act to cases involving patents are not far to seek. So long as monopolistic control is exercised only over articles or processes which are themselves patented, that is only carrying out the established purposes of the patent law, which gives to the owner of the patent for a limited term the absolute monopoly of the patented thing, to do with as he sees fit. The public is not so deprived of anything to which it was entitled before the patented invention was made, and self-interest may be relied upon to insure adequate exploitation of the invention if it has any real value. The only exception I can think of is where potentially competing patents are acquired and suppressed, and even in those cases the time when the suppressed invention becomes free to the public by the expiration of the patent is not deferred.

I think the cases in which that is done, when the suppressed patents could not be suppressed for the greater part of their term even in the hands of a competitor, by reason of earlier and controlling patents, are rare. And I very much doubt that the harm done is anything like as great as the harm which would result from the confusion and limitations of the rights of inventors, if the compulsory license system, which I have referred to in connection with foreign patents, should be adopted in this country.

On the other hand, patents have unquestionably been used in this country to restrain trade in articles which are not in themselves patented. To take a prominent example, it has been and still is the practice of some manufacturing concerns who control by important patents articles which, as a practical matter must necessarily be used by certain classes of consumers, to refuse to sell such articles unless the consumer agrees not to buy from anyone but the manufacturing concern in question certain other articles of related character, but which are in themselves not patented and which but for such agreement could be freely bought and sold by anyone. Personally I have no doubt that this practice, and all similar practices which unduly restrain trade in unpatented articles, are contrary to public policy and are prohibited by the Sherman Act. It is true that the courts have sustained the right of an owner of a patented machine to license the use of such a machine coupled with the condition that the material used in operating the machine will be purchased from the licensor, but a little examination of that situation will show that the restriction does not extend beyond the limit of the patent monopoly because it does not restrain in general the purchase and sale of such materials, but merely restrains the use of the patented machine. The licensee is entirely free to buy such materials for other purposes, and if he does not want to use the patented machine, to buy them for any purpose.

I think, therefore, that your duty toward the patent system is to treat it fairly when you have to deal with it at all; to respect the patent rights of others and insist upon your own; to investigate the criticisms of it in a spirit of friendliness and not to repeat them in a spirit of hostility unenlightened by such investigation; and if you are ever called upon to testify as experts in patent cases to lay aside all jealousy and bias, and to state the facts as you know them to have existed at the date of the supposed invention as fairly and as lucidly as you can.

SPEEDS AND COST OF DIAMOND DRILLING AT PORCUPINE, CANADA.

BY ALBERT E. HALL.*

In any discussion of this subject, it must be realized at the outset that local conditions play a large part. For example, consider the water for the drills: In one case a drill was set close to a spring, so that the cost was very small; in another case the water had to be obtained from a lake 2000 ft. distant. A pump could not be used, as the water froze, so an injector was installed. The cost in this case, where two drills were using the water, was 35 c. per foot of hole, and had there been but one drill the cost would have been 70 c. per foot. These two cases illustrate extremes in the procuring of water, and similar variations are to be found in other items. No fair comparison of costs can be made, therefore, unless all the conditions are known.

The price of carbons varies widely, whence quotations must be known to be able to compare costs at different periods. The following table gives the cost of carbons at intervals since 1875:

PRICE OF CARBONS PER KARAT.

1875	$ 10	1906	$ 78
1893-4	15	1908-9	80 to 95
1896	20 to 22	1910, 1 to 1½ K.	40
1897-8	35 to 36	1½ to 2 K.	55
Bortz	25	2 to 3 K.	65
1899-1900	70	Bortz, 1 to 1½ K.	8
1901	36 to 60	" 1½ to 2 K.	12
1903	$ 50	" 2 to 3 K.	14

A machine whose operating cost per foot in 1893 was $0.75 might appear to have been doing work cheaper than another which gave a cost of $3.00 per foot in 1910. Assuming the consumption of carbons to be the same, however, such is not the case, as may be seen by the proportion, $15 : $60 :: $0.75 : $3, taking into consideration the advancing price of carbons.

In our drilling operations at Porcupine, one machine made 900 ft. of hole in 26 working days, the last of which we have record. The work was done in two ten-hour shifts per day. The rock

* E. M., Columbia, 1911.

was schistose, containing quartz veins and stringers. The inclination of the hole was 54°, while the formation dipped at 73° in the opposite direction. Following are the itemized costs per foot:

Management and diamond setting	$0.167
Illumination (kerosene lamps)	0.100
Coal	0.295
Lumber	0.050
Casing pipe	0.010
Freight	0.100
Blank bits	0.010
Core barrel and shell	0.010
Labor: Runners, $100 per month Firemen, $2.75 per day Core checkers, $2.75 per day	0.521
Wear of carbons	0.175
Water	0.050
Depreciation of plant	0.005
Depreciation of carbons	0.016
Total	$1.509

In the case of another drill working in the same formation and under the same conditions, the costs, based on 1163 ft. of hole drilled in 34 working days, amounted to $1.566 per foot. This includes the sinking of a stand-pipe to a depth of 30 ft., which took three days. Coal costs $7.00 per ton laid down at the drill.

Many engineers and drill owners lose sight of some of the items, and imagine that they are getting very low costs. Engineers take account of the wear of diamonds, but often neglect the depreciation of the stones. For example, eight stones of 2½ karats, at $85 per karat, would have a value of $1,700. Suppose 5,000 ft. of hole drilled with these stones, after which they are found to weigh only 12 karats. The loss of eight karats at $85 amounts to $680. Many operators stop here, charging the item to wear of carbons. This is not enough; the stones now weigh only 1½ karat apiece, and hence their value per karat has depreciated, in addition to the actual loss of weight. If the worn stones be taken at $70 per karat, their total value is $840; this leaves $180 unaccounted for, thus: $1,700—(680+840)= $180, which must be charged to depreciation.

Following is the itemized cost of moving and setting up one of these machines. The shed was built for winter service, and was fairly substantial:

Tearing down old shed, ½ day:		
1 setter at $5.00	$2.50	
2 runners at $3.50	3.50	
2 firemen at $2.75	2.75	
2 core men at $2.75	2.75	
		$ 11.50
Moving materials to new location, 1 day:		
The crew at $17.50	$17.50	
2 core checkers at $2.75	5.50	
1 team at $1.50	1.50	
1 driver at $2.50	2.50	
1 helper at $2.50	2.50	
		29.50
Erecting new shed and machine, 2⅔ days:		
Lumber, nails, tar paper, etc.	$40.00	
The crew at $17.50	40.83	
2 core checkers at $2.75	12.80	
		93.63
Total		$134.63

The data in the following tables refer to holes drilled in schistose rock with quartz veins and stringers. The holes were of 1⅝ in. diameter, and were bored at an angle of 54°. The following table shows the speeds:

SPEED OF DIAMOND DRILLING: 1⅝-IN. HOLE.

	Theoretical; no allowance for pulling rods, etc			Actual; including pulling of rods, adding rods, etc.		
	per hour	per shift	per day	per hour	per shift	per day
Maximum	5.4	54.0	108.0	5.0	36.0	71.0
Minimum	2.9	29.0	58.0	1.0	14.0	26.75
Average	3.75	37.5	75.0	1.7	17.5	35.0

The first half of the above table shows what the speed would be if all the time were consumed in drilling; the figures in the second half are from actual observation, whence the time required for handling the rods and for making necessary repairs can be deduced.

The following comparison relates to two shifts' work:

SHIFT NO. 1	SHIFT NO. 2
Depth of hole, 1100 ft.	Depth of hole, 1155 ft.
Speed of drill, 700 r.p.m.	Speed of drill, 1000 r.p.m.
Started drilling with 3 ft. of core in the barrel.	Started drilling with 1 ft. of core in barrel.
6.45- 9.20, Drilling.	6.45- 8.36, Drilling.
9.20- 9.50, Testing for block and reblocking.	8.36- 8.45, Removing 5 ft., adding 10 ft. of rod and running back feed.
9.50-10.38, Pulling rods.	Speed of drill changed to 700 r.p.m.
10.38-10.42, Repairing bit.	8.45- 9.35, Drilling.
10.42-11.12, Lowering rods.	9.35- 9.38, Running back feed and oiling.
11.12-11.16, Washing hole and starting.	9.38-10.33, Drilling.
11.16-11.40, Drilling.	10.33-10.36, Running back feed.
11.40-12.00, Repairing engine.	10.36-10.53, Drilling.
12.00- 1.00, Dinner.	10.53-11.10, Adding 5 ft. of rod, testing and adjusting blocking twice.
1.00- 1.42, Repairing engine.	11.10-11.35, Repairing engine.
1.42- 5.15, Drilling.	11.35-12.35, Dinner.
5.15- 5.28, Repairing gears.	12.35- 1.30, Pulling rods.
5.28- 6.22, Pulling rods.	Hole was stopped at this point.
Left rods up at end of shift.	

Contrary to expectations, holes dipping with the formation seem to give better core than those that dip across the formation. In the first case, the recovery is 95 to 100 percent., while in the second case it is between 90 and 95 percent. In drilling parallel to the dip, single pieces of core 10 ft. long have been obtained with a core barrel 10½ ft. in length.

December 10, 1911.

A STUDY OF MECHANICAL CONVEYORS.—II.*

BY LINCOLN DeG. MOSS.

Belt Conveyor: Evolution; Lubrication; Composition of Belts; Balata Belts; Pulleys; Trippers; Power and Efficiency.—Aerial Tramway: Evolution; Capacities and Efficiency.—Cable Railways: Grades and Curves; Grips; Cables; Drivers.

BELT CONVEYORS.

During the war with the Seminole Indians, biscuits for the troops were baked at St. Augustine on a machine invented by W. R. Nevins of New York City, who employed an endless canvas belt conveyor running on flat idlers. During the 60's, the Choice & Gibson brickmaking machines employed flat, endless belts to carry molded bricks from the pug mills to the stacks for baking. In the same decade, Cook's magnetic ore separator was used at Plattsburgh, N. Y., for carrying ore from a vibrating hopper under a revolving magnet which left the rejected material on the belt to go to a waste pile.

Some eminently practical work was done by Lyster at the Liverpool (England) docks, described in the *Proc. Mech. Engr.*, August. 1869. Finding that spiral conveyors in troughs lacked capacity and used a large amount of power, he experimented with endless bands. He developed several essentials, such as the "throw-off carriage," or tripper, so arranged that the dumping pulley could fold down and stop the throwing off of grain; a tripper which could discharge right, left, or back on the belt; the tension take-up; an endless, rectangular circuit with loading chutes and trippers on upper and lower levels. He also used rotary cleaning brushes and an air-blast brush. His tests showed that the helical conveyors required four times as much power to carry 50 tons per hour 100 ft. He first used concentrating rollers, Chart 15, Fig. 1, to increase the carrying capacity of the belt, which had two plies of canvas covered uniformly with vulcanized rubber. He abandoned this carrying idler, using thereafter 6-in. straight, turned, wooden rol-

* Extracts from lectures before the Department of Mechanical Engineering of Columbia University.

lers, because the difference in circumferential speed at the body and rim of the concentrator roll caused a slip at the rim which wore out the belt edges. A point on his belt, which moved 8 ft.

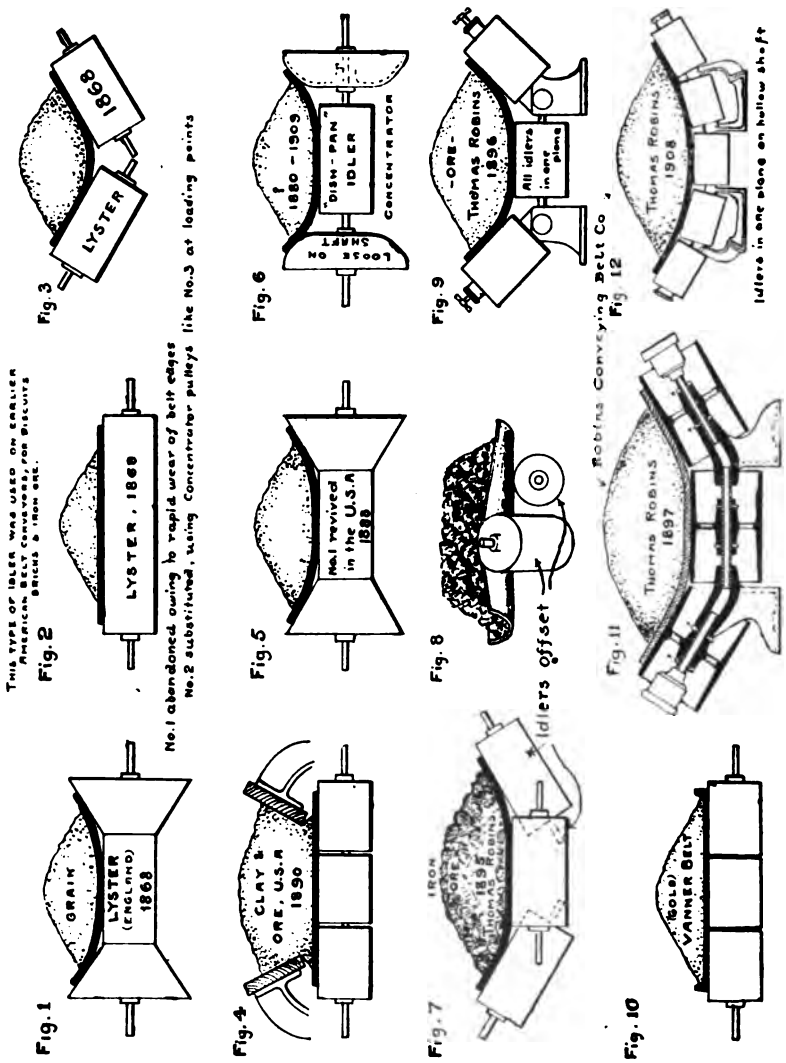


CHART 15. DEVELOPMENT OF BELT CONVEYOR PULLEYS.

per sec. for 10 hours per day and 300 days per annum, would travel yearly over 86,000,000 ft. The central, 6-in. diameter of the roller governs, as it carries the major part of the load. At the 9-in.

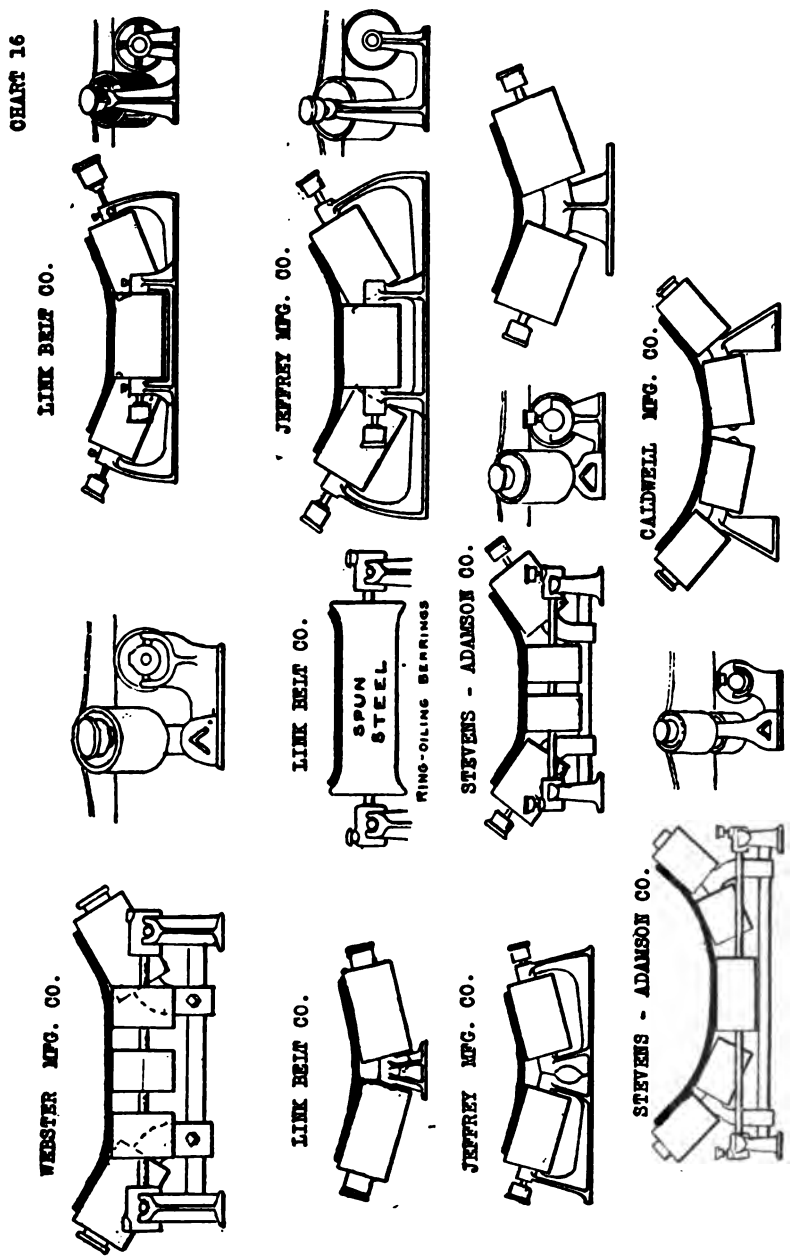


CHART 16. TYPES OF BELT CONVEYOR PULLEYS.

part of the cone, the slip would be over 43,000,000 ft., which explains why his belt edges wore out. With the flat, cylindrical idlers used thereafter, which was the older American practice, he used two diagonal rollers, Chart 15, Fig. 3, to concentrate the grain in the center of the belt at loading points. Thereafter grain was carried by belt conveyors almost universally.

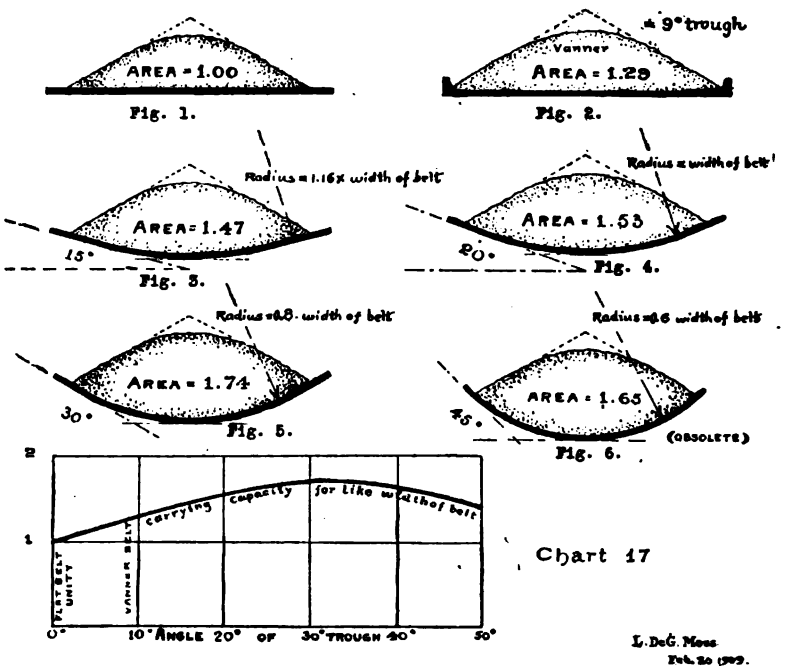
In this country, the Webster Manufacturing Company carried on such work for many years. In one of their large elevators, 3,500,000 bu. of wheat can be stored, requiring over two miles of conveying belts. Early in the 90's, iron ore was carried in the Edison mines of New Jersey on belt conveyors. Scraper conveyors had been abandoned owing to rapid wear. Belts of cotton, and of cotton with a rubber cover were used. Concentrator idlers were tried and abandoned, as in England. Flat belts with continuous skirt boards followed, with this fault: Particles of ore got between the belt and the skirt board near the center of a span between two idlers, where the belt sagged under the load; these particles, lodging, cut the belt edges, in time destroying them when the skirt boards were moved in to accommodate the narrowed belt. When skirt boards are used at chutes, their lower edges should be a thin sheet of steel which will not permit a lump to remain long.

Next we note the interesting experiments of Thomas Robins at the Edison plant. Not knowing the researches of Lyster, he learned the same truths. He tried two inclined concentrator rolls, and found that the belt did not "steer" true, but rolled into the apex of the angle at the two pulleys, cutting the belt. The next step was to add a third pulley set horizontally and in the wake of the two concentrators. Chart 15, Figs. 7-8. Oil lubrication gave trouble, as some oil threw out centrifugally on the belts and ate holes in the cover. This must be carefully avoided when using idlers lubricated by oil. Although friction is greater with grease lubrication, Mr. Robins adopted it thereafter, using fixed hollow shafts, with the opening at the center of bearings. This formed collars of grease at the ends of the journals and made a perfect dust guard in the worst situations. Not only is this desirable for idlers, in such service, but it is excellent for the main bearings of head and tail rollers, using solid shafts. Several very effective oil lubricated idlers are now made, with dust guards.

The belts at first had a cover of uniform thickness, top and bottom. Mr. Robins next built his belts with a thicker carrying cover. This belt is now patented and is in very general use. Then noting that these new belts wore away faster in the center, he made a still thicker center, either straight or tapered to the edges. He next made a belt like Chart 17, Fig. 2, which was very flexible in cross section and had stiff edges to support the catenary between two idlers. His idlers finally were built in one plane, like Chart 15, Fig. 9. The angle of inclination of the wing ilders was 45°. This caused too great a flexure on the belt and actually reduced its carrying capacity, which is maximum at about 30° inclination. (See Chart 17.) An angle of 30° is the maximum used at present, although some old installations using steeper angles are very successful. Troughing a belt has a commercial bearing by increasing its carrying capacity and decreasing the wear caused by the materials. A given quantity of material loaded on a wide belt will be shallower in depth and will cause more wear than when a narrow layer is applied and the rest of the load superimposed upon the first protecting stratum. Troughing such as can be secured by the natural flexure of a belt, without forcing, is not injurious. Belts of ordinary uniform construction will give a considerable natural trough when 24 in. wide and upwards. Heavy belts are naturally stiffer than thinner ones.

A rubber belt is not homogeneous, being built up of a number of plies of cotton duck cemented together, under heavy pressure, with rubber. Makers call this the "friction." The built-up plies are finally covered with rubber, and vulcanized under pressure. Much of the success of a belt depends on the "friction" between the plies. The purer the rubber used, the better the adhesion. This is usually tested by cutting a ply loose, 1 in. wide, hanging the belt vertically and attaching a weight to the loose strip, adding to it until the weight will just peel the strip loose. This value ranges from 7 or 8 lb., with inferior belts, up to 26 lb.; 16 lb. is excellent. As the belt bends over either end or tripper pulleys, or transversely by troughing, forces tending to separate the plies are set up, resisted by the elastic, tenacious rubber between the plies. Bending over pulleys is of more consequence than the bend of troughing, as the radius of pulleys is much less than the radius

of the trough, and a belt is stiffer along its length than transversely; this is due to the construction of the canvas, which has long continuous longitudinal fibres, unbroken for as much as 400 ft., while the transverse fibres and threads are short, not quite as long as the width of belt, fewer in number, and disconnected. The wearing cover, for commerical reasons, is compounded with other materials to lessen cost. Success lies in skillful compounding, which will not depreciate the wearing quality to a marked extent. The percentage of adulterant can be found by burning a strip of rubber,



and noting the residual ash; pure rubber leaves none. A good friction test is valuable. The use of high-grade, long-fibred cotton duck adds tensile strength and increases cost. Tensile strength is not a prime requisite, especially at the high speeds of transmission commonly adopted. Some experts judge the quality of rubber by its taste, but that cannot be formulated. It is safer to trust a reputable maker whose reputation is at stake, than to specify how he shall make a belt, for in case the durability is poor, and the consumer has specified details of construction, he becomes more or less liable.

The durability of a belt depends much on the manner of loading and operating; another factor is length. For a given output and speed, a spot on a belt 100 ft. long will come under a loading chute twice as frequently as on a belt 200 ft. long, and sustain twice the wearing effect of loading. If the chute delivers the material at a fair velocity, in the direction of the belt travel, the wear of loading will be reduced. If the material is fed in a thin, intermittent stream, the rubber cover will suffer much more than if the load is fed on continuously and the belt trough kept full. In the latter case, only a small percentage of the material ever touches the belt. Hence the important rule that a belt should be so proportioned to its load that it will be kept full, and operated at a speed which will require it to run full. When belts are installed with reference to future increase of carrying capacity, it is better to operate at first at a low speed, and then change the gear ratio for higher speed at the larger output. If the material carries a fair percentage of fines, such as crushed ore, it is well to arrange chutes as made by Thomas Robins in his early work in conveying iron ore. These chutes had grills or screens which dropped the fines on the belt, making a bed over its carrying surface to receive the large chunks. Badly arranged deliveries may destroy a belt in less than a week. The best record for durability which has come to my notice is at Louisburg, Canada, in the ship loading trestle of the Dominion Coal Company. Coal is accumulated in a 5,000-ton pocket, by railway gondolas, and is drawn off by a series of 36-in. Robins belt conveyors which elevate the coal to loading spouts which fill the hold of a 3,000-ton collier in four hours. The belts run at about 700 ft. per min. The coal is bituminous with few lumps. The present belt has carried over 3,000,000 tons and is good for more service. For steady, hard service we quote the conveyors of Jones & Laughlin, which supply their Eliza furnaces at Pittsburg, Pa. The record for one year showed that the conveyor ran for 363 days, 9.2 hours per day, and transported 500 tons per hour. This is a Robins installation.

For a wide range of service, the plant of the New Jersey Zinc Company, at Franklin Furnace, N. J., uses over 200 belt conveyors which handle franklinite (a trifle lighter than galena) from man-size rock down to finely crushed sands, which pass through Wether-

ill magnetic separators. Many of these conveyors operate in clouds of heavy, gritty dust, which shut out view at a distance of 3 ft. The conveyors are Robins. Those used in the magnetic separation run on bronze pulleys. The hot sands are carried on canvas belts; the wet sands and rock are moved on rubber and canvas belts.

In the vicinity of New York you may see Link-Belt belt conveyors at the 74th St. station of the Interborough Company and coke conveyors, made by the same concern, but using belts with little trough running on pressed steel idlers with ring oiling lubrication. (Chart 16.) Stephens-Adamson belt conveyors are used at the new Pennsylvania terminal. Robins conveyors are in use at the Subway power-house, 58th St. and Hudson River, at the Kingsbridge power-station on the Harlem river, and at the plants of the Consolidated Gas Company, at Astoria and 138th St. and East river.

For coal, a life of 1,500,000 tons may be anticipated. For ores, 500,000 to 1,000,000 tons; sand, ashes and coke follow in that order, coke being the most destructive. The belts are not suited for materials hotter than 150° F., nor for carrying oily substances. They resist water, gases, alkaline or acid ashes, wet cement and common salt. Many operate entirely exposed to the elements, in hot and cold climates. They resist abrading materials to a remarkable degree. They are not entirely waterproof, for the following reasons: Assume a dredge belt carrying wet tailings of gravel and stone. Fibres of the outer plies of cotton duck project up into the rubber cover, which is put on in a plastic state. The cover, when new, would be $\frac{3}{16}$ to $\frac{1}{4}$ in. thick. As it wears down, these small fibres of cotton carry water into the body of the duck, setting up rot and separation between cover and canvas. Some sharp pointed stones may penetrate the cover down to the canvas and hasten this action.

In the South African mines, considerable balata belting is used for handling wet materials. The belt is composed of cotton duck cemented by balata gum applied at the consistency of milk, under heavy pressure. No vulcanizing is necessary. The heaviest duck used with rubber belting is 32-oz., open weave, to permit the plastic rubber to pass through and bond. With balata belting, a 36-oz. duck, closely woven, may be used because the thinner, milk

like juice will pass through more readily. For this reason, and also because no heat is used, which diminishes the strength of duck, balata belts are relatively one-fifth stronger. Several oils will dissolve balata gum, which is obtained mainly from the Dutch East Indies. The proper mixture of solvents, and their elimination after the belt has been made, are trade secrets. It would seem that a belt having a homogeneous, waterproof, balata and canvas core. with a wear resisting rubber cover, would be an ideal conveying belt, provided that a stout bond can be made between the two materials.

The spacing of the troughing idlers should bear a relation to the weight of material on the suspended span between two adjacent idlers. The minimum spacing for heavy ores is 3½ ft.; for coal, 4½ to 5 ft.; for light materials, 6 ft. Return idlers carrying the slack belt may be set 10 to 15 ft. apart. The diameter of pulleys should bear a relation to the number of plies in the belt, just as we use certain ratios of rope diameter to sheave diameter. Other considerations enter. The larger the diameter of driving pulleys, the lower the speed becomes, necessitating a larger and more wasteful gear reduction between pulley and motor. It is often difficult to arrange a delivery from the head pulley to the tail of another conveyor, even with small pulleys, and get satisfactory chutes.

Driving pulleys are generally made larger than tail pulleys and tripper pulleys. For rope transmission, it is considered that all pulleys, whether drivers or carriers, should have like diameters, if the maximum life is to be obtained from the rope. At the driver, the extension of the fibres is that due to driving tension plus the extension due to bending around the wheel. Thousands of belt conveyors are running with pulley diameters of 4 to 5 in. per ply of belt. We consider that a ratio of 6 in. per ply would be preferable, if local conditions permit. A head drive is cheapest and most desirable as a rule. Tail drives are frequently used, especially when two conveyors are to be driven, one at its head, the other at its tail pulley, by a common motor. The journal pressure on the head shaft of a tail driven conveyor will be twice that of one driven at the head. Center and intermediate drives are much used, but require more mill work than head or tail drives. It is gen-

erally preferable to drive a boom conveyor from the tail end. For heavy driving, some very successful installations employ a modification of the old letter S American rope drive. The same principle is used in some Koeppe mine hoists, with two tandem rope sheaves. This drive was first used on a 36-in. Robins conveyor 1050 ft. long between centers, which had a stacking tripper connected with it. This machine was used for filling in swamp land at Riker's Island, New York City, with ashes and dry city refuse. The drive transmitted 80 h. p. at the tail end, and has since been widely used.

Driving power is measured by (1) Arc of contact; (2) Tension; (3) Coefficient of friction, in the same way that friction drives for Whiting or Koeppe hoists, cable drivers and traction elevators are determined, using Rankin's formulas for endless bands. The arc of contact may be increased by using a snub pulley behind the driver. In most examples, this snub pulley is entirely too small. The letter S drive will give twice the arc of contact obtained by a snubbed driving pulley. In driving cables with two drums, the leading drum performs over $\frac{2}{3}$ of the work. That is probably true of the letter S drive applied to belt driving. The coefficient of friction may be raised from 0.2 to 0.3 by lagging the pulley face with rubber belting bolted to the pulley face with countersunk bolts. Some slippery materials will decrease the coefficient of friction, the fine dust getting between the pulley face and the belt. Water also reduces the coefficient of friction, which explains why we have quoted 0.2 and 0.3 for designing. The working tension should not exceed 20 lb. per inch width of canvas per ply, nor should it exceed one-fifteenth the ultimate strength on account of the splices, which are made preferably by butting the belt ends together, holding the two ends by tension belt clamps, and connecting with Crescent or Bristol steel belt hooks. It is advisable to pierce the holes with a sharp tool before driving in the lacings, which are clinched on the inner side. Patches may be put in by the same method. Mr. Lyster in 1868 discarded pneumatic sweeping because it required much power and wore the belt; he therefore used a rotary brush, which was not altogether satisfactory. That holds true to-day. In moving damp materials, when

cleaning is most desirable, the brushes clog and do little effective work. A water jet spray has been used when moving wet concrete.

CHART 18. CARRYING CAPACITY IN TONS PER HOUR.

Materials weighing 50 lb. per cubic foot. Belts troughed to 30° from horizontal; trough section equivalent to 1.7 that of a flat belt. Also horsepower per 100 ft. of level conveyor. Grease lubrication. For each fixed tripper add 5%, but not less than 0.8 h.p. For moveable tripper add 6%, but not less than 1 h.p. For lifting on inclines, add Tons per hour × Feet lift ÷ 990.

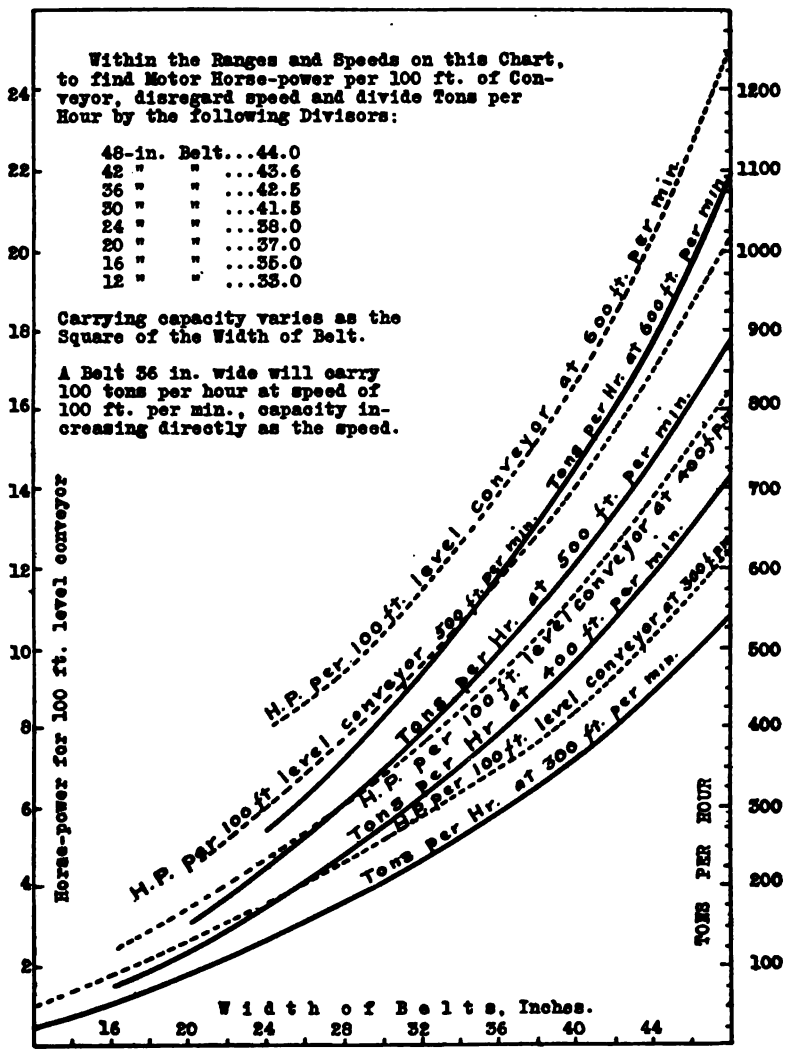
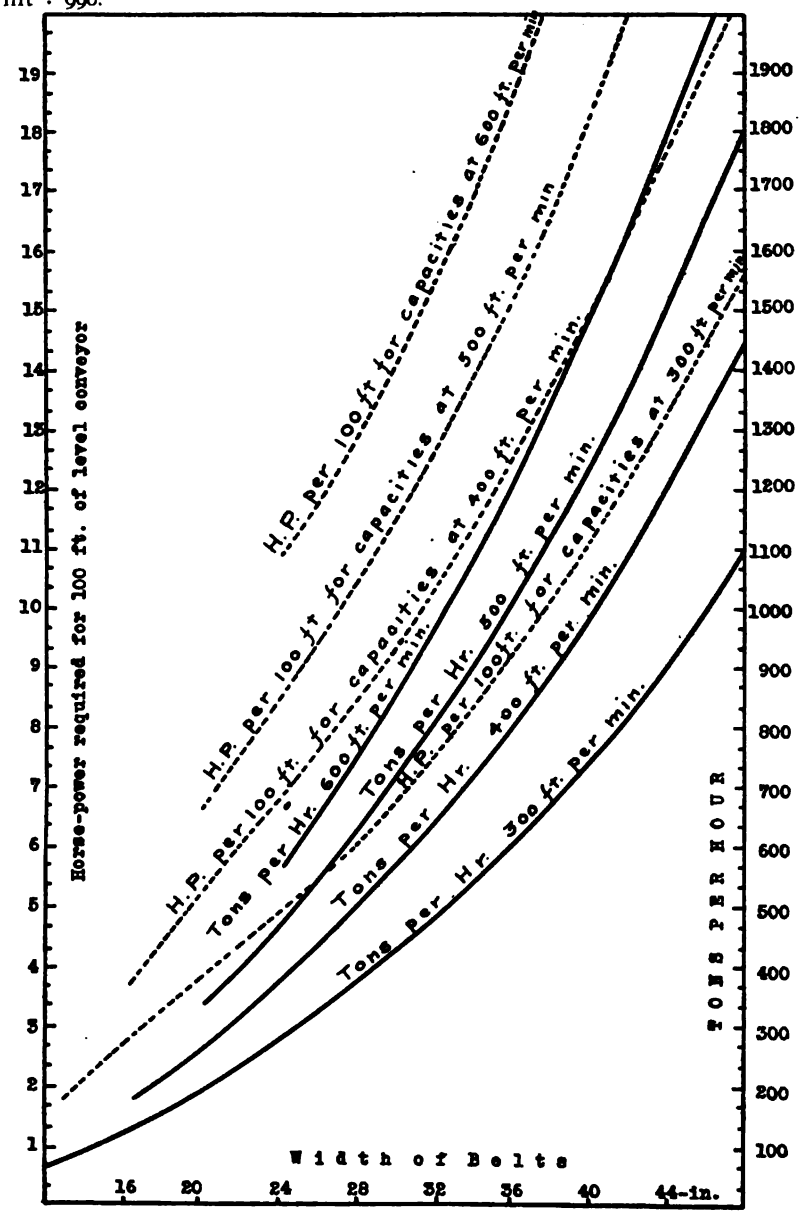

Within the Ranges and Speeds on this Chart, to find Motor Horse-power per 100 ft. of Conveyor, disregard speed and divide Tons per Hour by the following Divisors:
48-in. Belt...44.0
42 " " ...43.6
36 " " ...42.5
30 " " ...41.5
24 " " ...38.0
20 " " ...37.0
16 " " ...35.0
12 " " ...33.0
Carrying capacity varies as the Square of the Width of Belt.
A Belt 36 in. wide will carry 100 tons per hour at speed of 100 ft. per min., capacity increasing directly as the speed.
H.P. per 100 ft. level conveyor, 500 ft. per min.
Tons per Hr. at 600 ft. per min.
H.P. per 100 ft. level conveyor at 600 ft. per min.
H.P. Tons per Hr. at 500 ft. per min.
Tons per 100 ft. level conveyor at 400 ft. per min.
H.P. per 100 ft. level conveyor at 400 ft. per min.
Tons per Hr. at 300 ft. per min.
Horse-power for 100 ft. level conveyor
TONS PER HOUR
Width of Belts, Inches.
16 20 24 28 32 36 40 44
2 4 6 8 10 12 14 16 18 20 22 24
100 200 300 400 500 600 700 800 900 1000 1100 1200

CHART 19. CARRYING CAPACITY IN TONS PER HOUR.

Materials weighing 100 lb. per cubic foot. Belts troughed to 30° from horizontal; trough section equivalent to 1.7 that of a flat belt. Also horse-power per 100 ft. of level conveyor. Grease lubrication. For each fixed tripper add 5%, but not less than 0.8 h.p. For moveable tripper add 6%, but not less than 1 h.p. For lifting on inclines, add Tons per hour × Feet lift ÷ 990.


Horse-power required for 100 ft. of level conveyor
TONS PER HOUR
H.P. per 100 ft. for capacities at 600 ft. per min.
H.P. per 100 ft. for capacities at 500 ft. per min.
H.P. per 100 ft. for capacities at 400 ft. per min.
H.P. per 100 ft. for capacities at 300 ft. per min.
Tons per Hr. 600 ft. per min.
Tons per Hr. 500 ft. per min.
Tons per Hr. 400 ft. per min.
Tons per Hr. 300 ft. per min.
Width of Belts
16 20 24 28 32 36 40 44-in.
1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16 17 18 19
100 200 300 400 500 600 700 800 900 1000 1100 1200 1300 1400 1500 1600 1700 1800 1900

Trippers may be fixed, movable by hand, or moved by power derived from the belt. Fixed trippers require more power, in proportion to the number used, and are justified in few cases. For example, if colliers are to be loaded, taking coal simultaneously at bow, midship and stern hatches, a fixed tripper at the first and second fixed point and one end delivery for the third point would be the most economical arrangement, the first tripper taking off $\frac{1}{3}$ and passing $\frac{2}{3}$ over to the second tripper, which would take $\frac{1}{3}$ and pass the remainder to the end pulley. Over a boiler house bunker, however, with three fixed trippers, and dumping at one point at a time, the coal would be lifted over each preceding tripper and dropped back on the belt, causing needless wear and wasted power. A hand propelled tripper running on T-rails will answer in most cases, for as a rule trippers are not run back and forth to spread the material, but are clamped down to the rails to build a conical pile and then moved a few feet so as to deliver on the crest of the cone. The extra expense of automatic trippers is often unjustified. The framing of a tripper should be very stiff and substantial, for if elastic the two belt pulleys in the tripper will deflect and cause the belt to run to one side. To correct a tendency to run to one side along the line of the belt, the idlers should be arranged to be canted, by slotting the feet of the pedestals at the bolt holes. Side-guide idlers should be used as little as possible, to avoid wearing the belt edges. It is better to train the belt by canting the idlers and end pulleys.

Power, Capacity and Efficiency of Belt Conveyors.

Up to the present time these factors have not been well understood, for two reasons: (1) This form of conveyor has been in general use for heavy work, for little more than 12 years, since

NOTES ON CHART 19.

Carrying capacity varies as the square of the width of belt. At every speed in the range on Chart 19, with materials weighing 100 lb. per cubic foot, for every 100 ft. of level conveyor, the following horse-powers are required:

With	48-in.	belt,	H.P.=T. P. H.÷ 70.
"	42	"	H.P.=T. P. H.÷ 67
"	36	"	H.P.=T. P. H.÷ 64
"	30	"	H.P.=T. P. H.÷ 61
"	24	"	H.P.=T. P. H.÷ 54
"	20	"	H.P.=T. P. H.÷ 50
"	16	"	H.P.=T. P. H.÷ 50
"	12	"	H.P.=T. P. H.÷ 50

it was made practical for coarse and heavy materials by Thomas Robins. (2) The engineers concerned in its development have had little time to spare for analysis of the phenomena of this interesting transporter. To aid in clearing up these questions, I have prepared Charts 17, 18 and 19. When we cease to generalize, and group observations and data with care, consistent curves of performance, power and efficiency can be plotted.

For uniformity, I have referred efficiency back to the motor input, which can be measured by volt-ammeter at the motor terminals, and is equivalent, as nearly as possible to the indicated horse power of a steam engine. This seems more just than to disregard the friction loss in the gearing between conveyor driver pulley and the motor, and to omit the inevitable losses in the motor, as some writers have done. The great effect on efficiency due to varying density of the materials carried has been generally overlooked, which is also true of observations on conveyors carrying very light loads. One gentleman has stated that his conveyor consumes 90 percent. of the total power in friction; he was without doubt correct. For example, a certain conveyor has a 30-in. belt, 386 ft. long on its carrying side, moving 348 ft. per min. and carrying 40 tons per hour. It used 13.4 h. p. with no load, and only 14.0 h. p. when fully loaded, showing 96 percent. friction loss. Another gentleman has a 30-in. conveyor, 400 ft. long, moving 400 ft. per min. and carrying 600 tons of ore per hour. The light power was 15.3 h. p., which rose to 33.7 h. p. when carrying full load, giving a friction loss of about 46 percent. Each observer was correct. The power required to run a given conveyor with no load, is exactly the same, regardless of the quantity and density of the material to be fed to the belt, whether it be feathers or galena. But the friction loss in handling a load of feathers would be over 99 percent., and would fall to less than 40 percent. when carrying galena on the same conveyor.

The wide discrepancy in the efficiency of the two 30-in. conveyors referred to can be further understood by studying Charts 19 and 20. The performance of a 36-in. conveyor, 100 ft. long, at speeds up to 600 ft. per min. is shown, first when carrying materials weighing 100 lb. per cubic foot, and second, when carrying materials weighing half as much. For comparison, the perform-

ance of a 20-in. conveyor is shown just beneath, handling materials of the same weights per cubic foot.

The total power loss when handling materials weighing 50 lb.

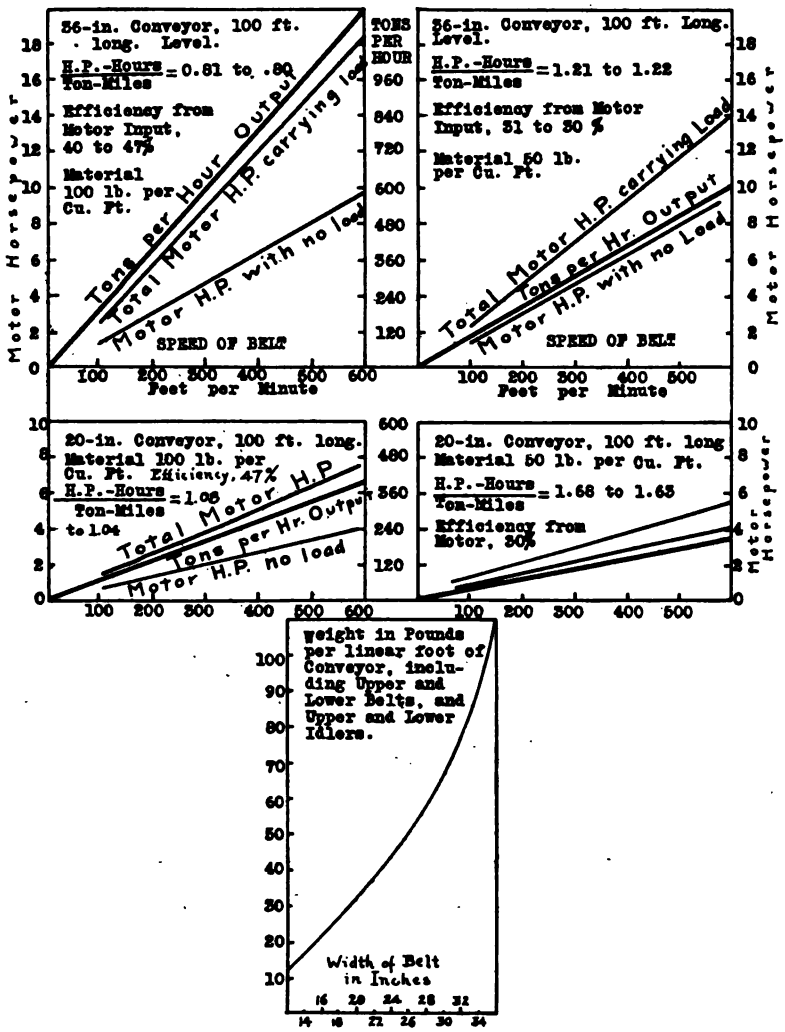
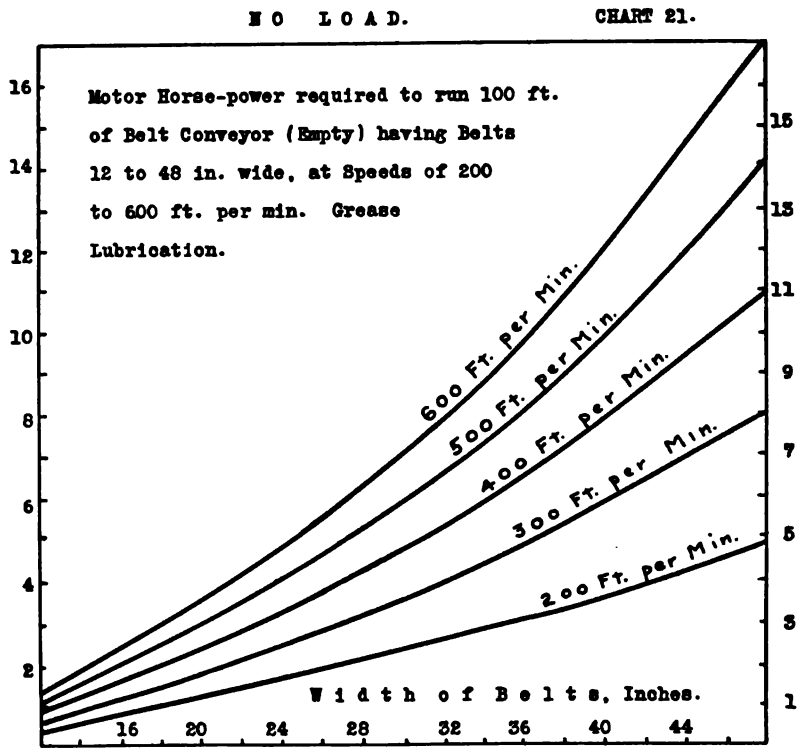

36-in. Conveyor, 100 ft. long. Level.
H.P.-Hours / Ton-Miles = 0.81 to .80
Efficiency from Motor Input, 40 to 47%
Material 100 lb. per Cu. Ft.
Tons per Hour Output
Total Motor H.P. carrying load
Motor H.P. with no load
SPEED OF BELT
Feet per Minute
TONS PER HOUR
Motor Horsepower
36-in. Conveyor, 100 ft. Long. Level.
H.P.-Hours / Ton-Miles = 1.21 to 1.22
Efficiency from Motor Input, 31 to 30 %
Material 50 lb. per Cu. Ft.
Total Motor H.P. carrying Load
Tons per Hr. Output
Motor H.P. with no Load
20-in. Conveyor, 100 ft. long. Material 100 lb. per Cu. Ft. Efficiency, 47%
H.P.-Hours / Ton-Miles = 1.06 to 1.04
Total Motor H.P.
Tons per Hr. Output
Motor H.P. no load
20-in. Conveyor, 100 ft. long Material 50 lb. per Cu. Ft.
H.P.-Hours / Ton-Miles = 1.68 to 1.65
Efficiency from Motor, 30%
Weight in Pounds per linear foot of Conveyor, including Upper and Lower Belts, and Upper and Lower Idlers.
Width of Belt in Inches

CHART 20. EFFICIENCY OF BELT CONVEYORS.

per cubic foot is about 70 percent. for either 36-in. or 20-in. conveyor. Of this, about 15 percent. is lost within the motor, and 15 to 20 percent. more in the gear reduction between motor and driver pulley of the conveyor. The gear-reduction loss depends

on the quality of materials, proper proportions, with accurate lining and leveling of boxes, and the gear ratio. If the driving pulley is large, the belt speed low, and the motor speed high, with bearings not proof against grit, the gear reduction loss may reach 40 percent., and rarely falls below 15 percent. The conveyor by itself will show 30 to 50 percent. friction loss at the driver pulley.

These charts I hope will prove valuable to the profession. Their use will save time in calculations, which require cumbersome for-


NO LOAD. CHART 21.
Motor Horse-power required to run 100 ft. of Belt Conveyor (Empty) having Belts 12 to 48 in. wide, at Speeds of 200 to 600 ft. per min. Grease Lubrication.
600 Ft. Per Min.
500 Ft. Per Min.
400 Ft. Per Min.
300 Ft. Per Min.
200 Ft. per Min.
Width of Belts, Inches.
16 20 24 28 32 36 40 44
2 4 6 8 10 12 14 16
1 3 5 7 9 11 13 15

mulas, made in some cases from incomplete and inconsistent data. Carrying capacity may be found by simple inspection, for belts from 12 to 48 in. wide, at speeds up to 600 ft. per min. For a given belt, capacity varies directly as the speed, and in direct proportion to the increase in weight of material per cubic foot. For different widths of belts, capacity varies as the square of the width. The motor horse-power per 100 ft. of level conveyor may also be found by inspection. Allowance for grades may be added or de-

ducted by the formula: T.P.H × Ft. difference of elevation of receiving and discharging end ÷ 990. Allowance for trippers is made as noted on the diagrams.

The load-carrying cross section is taken at the most economical troughing which can be obtained by a troughed belt, viz., 1.7 times that of a flat belt of equal width. (See Chart 17.) If flatter troughs, of less carrying capacity, are used, reduce the tons per hour shown on Charts 18 and 19, in proportion, referring to Chart 17. The horse-power will not fall in proportion, because a reduction in carrying capacity of trough does not affect the fixed friction at no load, and it is generally necessary to use a size larger belt to carry the required tons per hour. Data on conveyors having belts wider than 36 in. are meagre, but when properly used fall into the curves as shown. One exceptionally wide belt is in use, a 60-in. Robins, used as a picking belt for separating barrels, mattresses, bed springs and coarse waste from city refuse.

Designers will find some handy short cuts to results, noted on the charts. Especially interesting is the fact, which could not have been discovered without these performance curves, that the power required to operate a given conveyor, 100 ft. long, at any speed up to 600 ft. per min., may be found at once by using a divisor into the tons per hour to be carried. These divisors for different widths of belts are noted on Charts 18 and 19.

By applying the ton-mile method of comparison, we find the simplest of all short cuts. To find the power required to move a given tonnage, we may ignore the size of the conveyor and its speed, and simply allow 0.8 to 1.2 h.p.-hour per ton-mile, when the material weighs 100 lb. per cubic foot. For material weighing 50 lb. per cubic foot, allow 1.2 to 1.8 h.p.-hour per ton-mile. The smaller figures apply to the largest conveyors, the higher constants to smaller conveyors.

	Large conveyors	Small conveyors										
T.-M. in 10 hr.	×	0.08	to	0.12	=	motor h.p.,	materials	100 lb.	per	cu.	ft.,	at any speed.
T.-M. " 10 "	×	0.12	"	0.18	=	" "	"	50 "	"	"	"	" " "

To lift a load diagonally, belt conveyors may be inclined generally up to 21°. At steeper angles, there will generally be a small slip backwards which is injurious to the rubber cover. On grades, an unbroken stream should be maintained if possible. Coal is

elevated in one case on an angle of 27° by keeping a steady flow from the hopper, which prevents the coal from slipping down. Cement fresh from the mills cannot be carried up an incline unless it is first passed through a short screw conveyor which expels the entrained air. The most conspicuous use of belt conveyors as elevators is in the New York Subway power-house, where coal is elevated by a series of Robins belts, arranged on the switchback principle, to the monitor over the bunkers. The shortness of these switchback conveyors, compared with the long tunnel conveyor and the long distributing conveyors over the bunkers, makes a noticeable decrease in their belt life, for reasons stated earlier. In an ore mill which runs night and day, a 30-in. Robins elevating conveyor is in use for elevating 500 tons per hour of very heavy, sharp ore, reduced to coarse sand weighing 125 lb. per cubic foot. It contains a percentage of freshly broken quartz, very sharp and angular. This displaced a pair of first-class bucket elevators, which gave much trouble owing to the large capacity, high strains, lift of 80 ft., and the rapid wear by the very abrasive sand. The angle is 23°, which should not have exceeded 20° as there is an injurious slip on the belt cover. In this case the fall of a heavy bucket elevator meant a serious and expensive delay, whereas the belt gives signs of distress long before it will fail, and even if the belt pulled apart, the trouble would not be serious. Generally, we believe that it is better to elevate by bucket elevator and distribute by belt. Belt conveyors have sometimes been improperly used as elevators. Elevating boom conveyors with 30-in. belts, on booms 80 to 90 ft. long, at angles of 20 to 22°, are in general use on Western gold dredges, for carrying away the wet gravel tailings. A 36-in. Robins belt on a Danube dredge carries 1,000 metric tons of gravel per hour. A barge with a reversible pivoted boom conveyor was supplied by the Link-Belt Company for distributing coke for the Consolidated Gas Company of New York City. Another similar machine was built by the Webster Manufacturing Company for handling limestone on the Great Lakes. When using the charts for estimating carrying capacity, allowance must be made for the size of lumps in the material. Mine-run coal from the West Virginia fields frequently contains lumps as large as a man's body, requiring a 36-in belt. Ordinary mine-run bituminous coal has been suc-

cessfully carried on 24-in. belts. Foundry coke requires a belt 30 in. wide, and ordinary coke a 24-in. belt. Belts for grain require about 4 h.p.-hours per ton-mile because the material is so light. An 18-in. belt will carry 50 tons per hour and will require 3.8 h.p. per 100 feet.

Structures for belt conveyors are light, owing to the lightness of the conveyor. (See Chart 20.) Trippers with stiff cast-iron frames range from 1,000 to 3,000 lb. When trippers are used on trussed spans, do not forget to put in counters for the rolling concentrated load. Lateral stiffness is very important, to keep the idlers in line; otherwise the belt will not run true. Vertical deflections are of no consequence as regards the running of the belt. A continuous, dirt-tight deck over the return belt is very desirable. This may be worked out into a very stiff system of lateral bracing. A low house with hinged inspection doors is an economical method of covering. A walk alongside, for greasing, inspection, and cleaning, is required. Many belt conveyors operate outdoors in all extremes of climatic conditions, with no covering. On exposed trestles, high winds may blow the light belt out of the troughing idlers. To prevent this, a low wind-break fence has been built up on one side, in a few cases.

A conveyor with ball-bearing idlers has been introduced by the Conveying Weigher Company of New York. The diameter of the idler pulleys is ⅗ that of other makers, as the friction is less. A conveyor for carrying coal has been in use for two years, and shows a reduction in power consumption of one-third. The bearings receive no lubrication, being greased with vaseline when first assembled, and closed with felt washers.

Aerial Tramways.

Vehicles propelled by ropes are closely related to continuous conveyors, and yet possess intermittent functions. The single-rope system uses one endless running rope, carried over the supporting towers by revolving sheaves. The carriers are gripped to this rope. This rope also becomes the supporting span rope. It must be strong enough to sustain the vertical loads on the longest spans and to transmit the required pulling force. Therefore the rope must be of uniform strength and section throughout, and yet

be pliable enough to be driven. Flexibility requires ropes of relatively small individual wires, which do not possess the wearing qualities of ropes having larger outer wires made of higher-carbon steel. These facts limit the use of this type of tramway to spans of 600 ft., loads of 600 lb., and grades of 1 in 3. The simplicity of this system makes it the cheapest, within its limitations.

By using a fixed rope for trackway, and a separate traction rope, several advantages are gained. Very long spans may have large ropes which are too stiff to be commercially driven. The shorter spans may have ropes of reduced diameters to suit the spans. This track rope, on the unloaded side, may be lighter than on the loaded side. It may also have special interlocked wires on the outer wearing surface, made of high-carbon steel. The moving power rope may then be made very light and flexible. Grades of 1 to 1 may then be overcome; spans up to 2,300 ft. have been used, and loads up to 1 ton.

Aerial tramways, if longer than 3½ to 4 miles, are made in sections, shunting the buckets from one to the other. An Otto-Pohlig tramway in the mountains of Austria conveys timbers for 12½ miles, using four sections. This tramway crosses a ridge 800 ft. higher than the starting point, and has an average descent of 3½ percent. The system operates by gravity after the friction of rest has been overcome by a steam engine. A Leschen tramway built for the Penn-Wyoming Company at Encampment, Wyo., is 16½ miles long, in two sections, and carries 25 tons of ore per hour. A Trenton Iron Works cableway 22 miles long was built for the Argentine Republic to bring ore to Chilecito with a total fall of 11,500 ft. It carries 40 tons per hour, using buckets of ½ ton capacity. The cableway is in eight sections.

It is now evident that the principles of location are diametrically opposed to those governing the location of a surface railway. Balancing the descending against the ascending load through the endless rope connection eliminates the chief objection to grades. The shortest distance between loading and delivering points is usually possible by an "air line" instead of making wide detours to secure horizontal distance for easy gradients, which are essential in railway construction. These tramways are not competitors of the railway, but feeders thereto. Mining is possible in some cases only

by using this relatively cheap method of getting the ore down to a railway. They are sometimes used to increase the output over that obtained by animal transportation. In most cases, these roads originate in the mountains, often crossing ridges at a higher altitude and crossing chasms, rivers, and snow slides, on long, single spans up to 2,300 ft.

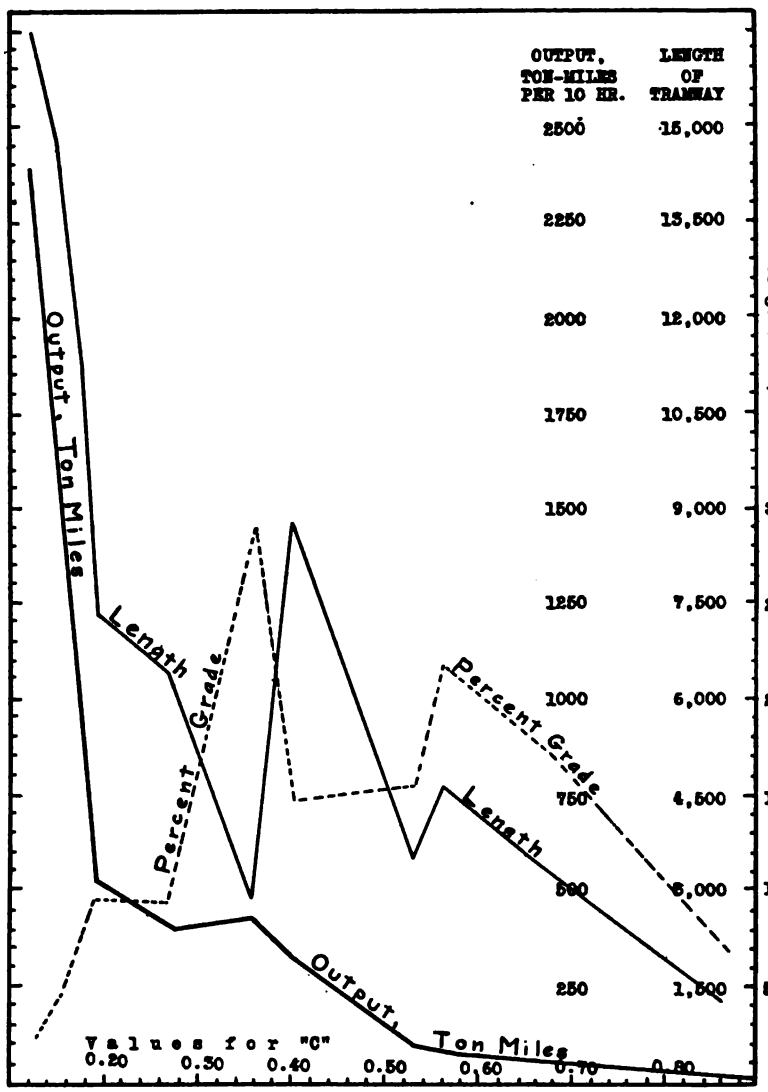
An average fall of 3½ percent. suffices to operate these tramways by gravity. In many places, a surplus of power is generated by the descending load, and is used to run crushers and elevators. In such automatic installations, one or two brakes are installed to control speed and prevent runaway accidents. The most economical tramway which I have yet noted is the "Split Rock Cableway" at Syracuse, N. Y. It delivers 750 tons of limestone in 10 hours to the soda-ash works of the Solvay Process Company. Its length is 16,500 ft., and average fall 239 ft., or not quite 1½ percent. In addition to the gravity pull exerted by the descending stone, 10 to 12 h.p. is supplied by a small engine. This is a Bleichert tramway, built by the Trenton Iron Works. It stands No. 1 in its class, for efficiency, and excels all other mechanical conveyors in the field which we are studying. The Split Rock tramway yields an output of 2,344 ton-miles per 10 hours with a power consumption which includes 11½ h.p. supplied by engine, and 18.1 h.p. supplied by the energy of 75 tons per hour falling 239 ft., a total of 296 h.p.-hours, which divided by 2,344 ton-miles gives 0.126 h.p.-hour per ton-mile.

Chart 22 could be prepared only after reducing the effects of up- and down-grades to a level basis, in the manner just described. It was compiled from data on existing tramways. It shows that long roads with high tonnage and moderate grades are the most efficient. It also shows that tramways with very steep grades suffer to some extent in efficiency, probably due to the higher journal pressures on the driving machinery due to the greater gravity pull of the load on the steep inclines, and to the use of heavier and stiffer power ropes. The mono-rail, suspension, lubricated track. and the balancing of the moving non-paying loads are great factors in the economy of this form of transporter.

Horizontal angles in the path of tramways are readily made. As in 1869, speeds remain 4 to 6 miles per hour, the lower speed

CHART 22. PERFORMANCE OF SOME AERIAL TRAMWAYS.

Showing Influence of Output, Length and Grade. The Base Line is laid out in Terms of Horsepower-Hours per Ton-Mile of Output. C = H.P.-Hours ÷ Ton-Miles. All Roads reduced to Level, by debiting Engine Power and crediting Excess Power realized. Grades are Average between Terminals.



being preferable. Tension stations are usually located at one terminus. On very long roads, they occur about one mile apart.

The grips of the carriers and tubs display much ingenuity. They are usually automatic in releasing and regaining their hold on the power line.

Cable Railways.

This means of transportation is closely related to the aerial tramway, but is applied to shorter hauls, usually less than one mile, with much lighter grades, but usually with considerable curvature in the track. Their capacity ranges from 60 to 300 tons per hour, depending on the number of cars on the circuit. The cars carry from 1 to 3 tons, and may be spaced, if necessary, 15 ft. apart; it is necessary to ungrip each car from the driving cable at the loading point, which should also be the point where the cable is bent down to pass to the friction driver. To reduce wear, it is desirable to keep the speed below 200 ft. per min.; 3 ft. per sec. is ideal. Some roads are speeded up to 250 ft. per min. when carrying peak loads, but it is better economy to use more cars at such times and keep the speed down.

Grades should not exceed 7 percent., because the cars are attached to the cable by friction grips. A man can set up a grip firmly enough to hold a car carrying 5,000 lb. on a 7-percent. grade. For steeper grades, a larger wheel must be used, about 30 in. diameter, set up by two men, or else cars carrying 1 ton must be fitted with the grip of a 2-ton car.

Curves of 12-ft. radius were introduced by C. W. Hunt, using his special running gear and track, one wheel running on its tread, the other on its flange. It is frequently impossible to use a larger radius owing to the limitations of available space. The point of tangent should never coincide with the point of reversed curvature ahead. A tangent as long as the total wheel base will permit the car and grip to straighten out before entering the next curve. Gauges of 18 to 24 in. are generally used. The Hunt standard is 21½ in. outside the rail heads, for outside flanges. Rails for 2-ton cars should weigh at least 16 lb. per yard; rails of less weight gutter the treads of the wheels and lack stiffness. Guard rails should always be used at the curves.

Cars are most durable when the bodies are made of seasoned oak and lined with sheet metal. Such bodies will last for 15 years,

the linings being renewed as necessary. Corrosion does more than attrition to destroy the linings. No. 18 to No. 10 steel sheets are used; the former will usually last for a year. Wheels should be not less than 12 in. diameter, made of steel, or with chilled treads if made of cast-iron. Springs are very desirable. Gable bottoms are desirable for a free delivery with side doors. The doors are drawn together by toggle, self-locking levers. A tapered chock on the track suffices to dump the cars by "breaking" the toggle.

A simple screw compression grip, with renewable steel blocks at the gripping contact, is the general favorite. The grip should be free to oscillate vertically, and be attached to the car by a radius bar, so that it may swing freely on curves, and be readily swung when detaching it from the cable. A plain grip must be held off the curve idlers by a continuous guard which should have a renewable steel wearing strip. The Hunt grip is made long enough to bridge three curve idlers and roll around the curve upon them, the grip sides being curved to the proper radius.

Curve idlers are usually 12 in. diameter, and overhung on the bearings. In a few cases the bearings are supported top and bottom. Idlers for carrying the cable on straight track should be as light as possible, and set not over 20 ft. apart, to keep the cable from chafing over the cross-ties. A broad face is desirable, 6 to 10 in. They should run easily, otherwise after the cable is lifted off by a passing car, and dropped back on the idler, the friction will cause the idler to lag, and the cable will score it. The logical idler therefore has as large a radius as possible with the least diameter of journal, and its weight should be kept down.

Cables for general service are crucible-steel hoisting rope, with six strands of 19 wires, laid up on a hemp core. Cables should be thoroughly soaked in oil and maintained in that condition. The stronger and harder grades of extra crucible-steel and plow-steel ropes are not adapted to this service. They require more power to run, and are hard on the grips and sheaves. A crucible-steel rope of $\frac{3}{4}$ in. diameter on a road 2,000 ft. long should carry 75,000 tons of freight. Records of 90,000 to 100,000 tons per rope are occasionally made. Light cable railways with cars of one ton capacity are run with $\frac{5}{8}$-in. cables. Drivers are made in three forms:

(1) Two drums with parallel grooves geared together by an equalizing gear, both drums driving. The equalizing gear is necessary because the leading drum performs over 75 percent. of the work of driving, as demonstrated by Mr. Leverich on the cable driver of the Brooklyn bridge. The leading drum therefore loses diameter more rapidly than the other. (2) Two drums, one driven, each with parallel grooves, the idler slightly tilted to feed the coils back to the driver. (3) One drum, not grooved but coned two ways from the middle of the drum face. The cone is not a straight taper but a curve with compound radii determined by experience. The cable takes four to six turns, leading on and off the drum from opposite sides. The cone causes the cable to work continually in towards the middle. Theoretically this lateral slip on the drum face is objectionable, but practically we have found no reduction in the life of the rope.

In the first and second systems, there is a strong thrust between the two drums, which requires heavy shafts and large bearings, with stout framing between. In the third system (C. W. Hunt Co.), the rope pressure is radial in all directions to the shaft centre, compressing the drum, with no thrust on shaft or bearings.

Cable drivers, Whiting and Koeppe mine hoists, traction elevators, and belt conveyor drivers, all depend for driving power on the same three elements: (1) arc of contact; (2) coefficient of friction; (3) tensions in the band. It is believed by some engineers that the coefficient of friction in drivers for wire ropes is not materially affected by lubrication, as the pressures are high enough to crush the lubricant. When a well oiled cable passes over a sheave, the oil in the core is squeezed out among the wires. When the cable straightens out, the oil is reabsorbed in the hempen core. Plenty of oil is a paying investment, reducing the power required to bend the ropes, a material item, and increasing the rope's life.

The cable railway sometimes takes forms which belong to the strictly intermittent type of carriers. It may consist of one or two reversible cars, permanently attached to a cable, and a reversing engine. In this form they are known as single or double shuttle cable railways. When two cars are used, one dumping while the other is loading, a double turnout is necessary to enable

the cars to pass at the middle of the run. Spring switches may be used, with two lines of rails. The "gauntlet" system is preferable, with four lines of rails, no switches, but two frogs. This was used on the Brooklyn bridge to reduce switching accidents, the trains going alternately to the south and north platforms. It is also standard with the C. W. Hunt Company for double shuttle railways. The speed may be run up to twice that of ordinary cable railways. With friction drivers, double tension take-ups are required, as the slack and taut sides of the cable alternate on each side of the driver. Check chains to limit the rise of the tension weights are obviously necessary. Deflecting sheaves, and those in the take-up loop, should have fully 48 in. diameter for ¾-in. rope.

Trestles for these railways are usually built of timbers, which are not acted upon by corrosion nor by the sulphurous emanations of coal; also, if exposed to internal fires in a coal heap, they char slowly where an unprotected steel column would soften and collapse. For these reasons, steel columns should be protected by red brick or concrete on wire mesh. Spans for timber construction are usually 15 ft. For protected steel columns, which are expensive, the spans should be increased, for economy, until the cost per linear foot of trusses equals the cost for columns per horizontal foot of structure. Avoid plank bracing on timber bents in piles of coal or stone. The sliding of the material will rip them off. Use framed and notched timbers, well bolted.

MODERN INCANDESCENT ELECTRIC LAMPS.†

BY LEONARD J. LEWINSON*

It is my purpose to limit this discussion to lamps employing filaments operating in high vacuum. This article comprises in part a review of published facts and data, the sources of which are listed in the appended bibliography. The general subject has been divided into several topics, arranged with a view to affording comparisons of the various types of lamps in present commercial use. These topics are as follows:

I.—Outline of the development of incandescent lamps with reference to methods of manufacture.

II.—Physical characteristics of present-day lamps.

III.—General discussion of efficiency, life and rating.

IV.—Classification of incandescent lamps according to their uses.

V.—Performance characteristics and data.

Commercial Types of Incandescent Lamps.

As early as 1842 experiments were made with platinum and iridium and their alloys as filament materials. These metals, however, were not sufficiently refractory for lamps of suitable efficiencies. Carbon-filament lamps were developed about 1878 to 1880. The early filaments were made from strips of woody plants, such as bamboo; from vegetable fibres, such as cotton threads; and from animal fibres, such as silk.[1] For example, bamboo strips were cut to the proper sizes, formed, and carbonized in a furnace. These filaments were found to lack uniformity in performance because of the structural irregularities which existed in the original fibres and, of course, remained in the carbonized product. A step in advance consisted in parchmentizing the outer portion of cotton threads, producing an irregular fibre which had to be drawn through a die. The early lamps manufactured from these various fibres were operated at from 5 to 6 watts per candle-power.

† Lecture before the Electrical Engineering Society of Columbia University.

* E. E., Columbia, 1904.

[1] For references, see Bibliography at end of this article.

A natural development of the parchmentized thread was the untreated filament, as it is now termed. The structural difficulties of the old fibres were avoided by the use of structureless cellulose. This cellulose may be obtained in a number of ways. Among the best known processes are the zinc chloride, nitro-cellulose and the cuprammonium.[2] Threads of cellulose produced by these methods are wound on large drums and dried. They are then cut and formed, and slowly heated in a gas furnace to about 550°C., at which temperature carbonization takes place. The baking is then continued, the temperature being rapidly increased to about 1,700°C. Filaments thus produced are operated in practice at about 3¾ to 5 w. p. c., depending upon the type of lamp and life desired.

The treated, or flashed carbon filament,[3] developed about 1893, was the next step. The base, or untreated filament referred to in the previous paragraph, is heated by the passing of current while surrounded by an atmosphere of hydrocarbon vapor, such as gasoline. The elements forming the hydrocarbon gas are dissociated and carbon is deposited on the surface of the base filament. This deposit, graphitic in nature, is of very much lower specific resistance than the base. The effect of the flashing process is to render the filament much more homogeneous and also to purify the base carbon itself. Lamps having treated carbon filaments are operated at about 3 watts per candle-power.

In 1905 the metallized carbon filament[4] was introduced. The manufacture is very similar to that of the carbon filament, through the base filament stage. This base filament is subjected to a high temperature in an electric furnace (about 3,000 to 3,700°C.), which does not appear to change the nature of the base carbon but is found to have certain beneficial effects on the final product. The filament is then flashed in air at about five times normal voltage for about one-fifth second, after which the usual "treatment" in hydrocarbon vapor is effected. Then follows a second electric furnace process, which causes the outer, or flashed portion, to undergo certain peculiar physical changes (referred to under "Physical Characteristics"), causing it to assume some of the properties of metals. This filament gives satisfactory performance at 2.5 watts per candle-power.

Carbon has a very high melting point — approximately 3,750° [5] — nevertheless it is impossible to operate carbon filaments at temperatures near the melting point because of the high vapor tension. The flashed carbon filament operates normally at about 1,800°. If the temperature is increased, the carbon vaporizes and deposits on the bulb, causing short life. This fact has been an important consideration in the development of metal-filament lamps, which employ materials less refractory than carbon, but capable of being operated at higher temperatures.

The first commercial metallic-filament lamp was the osmium lamp,[6] put on the market in Europe about 1903. The filament was produced by a paste process, the paste being squirted through dies into the required form, and the impurities driven out by subsequent heating. On account of the low specific resistance, a very long filament was required for even 110 volts, so that the lamps were made principally for voltages of from 30 to 60. The total available supply of osmium, however, is so small that it has been said that there is only enough for one year's supply of incandescent lamps. For this reason the lamps, which were used principally in Germany and Austria, were often loaned to the consumers and the osmium recovered after the lamp had burned out. These lamps were operated at about 1.5 w. p. c. In 1905 the tantalum lamp [7] was introduced, the filament being made from drawn tantalum wire. The filaments were comparatively long and supported on an arbor-like structure. On direct current the lamps give satisfactory service at from 1.8 to 2.0 watts per candle-power.

The tungsten-filament lamp [8] was put on the market very soon after the tantalum lamp, which probably accounts for the relatively small number of tantalum lamps in use in this country. A number of different processes for the manufacture of these filaments have been introduced. For some years most methods included the squirting of a substance containing tungsten particles through dies, forming thread-like materials from which the impurities were later expelled by various means. At first these filaments could be made in comparatively short lengths only. The principal processes were the paste process of Auer von Welsbach, the colloidal process [9] of Kuzel, and a deposition process,[10] used by Just and Hanaman, in which a carbon filament is heated in an atmosphere of a chloride

of tungsten and hydrogen. The tungsten is deposited on the carbon base, the carbon being eliminated in a subsequent step in the manufacture. In endeavors to make ductile tungsten, processes which involve the alloying of tungsten with other metals have also been developed.[11]

The principal objection to squirted filaments was their fragility. The inherent brittleness of the material combined with the unfavorable method of rigid mounting, rendered the lamps very delicate. One manufacturer introduced a continuous-wound, squirted filament[12] which, however, had to be wound about the supports while hot; the filament being loosely hung, and not rigidly fused to the supports, was naturally less fragile than the loop form.

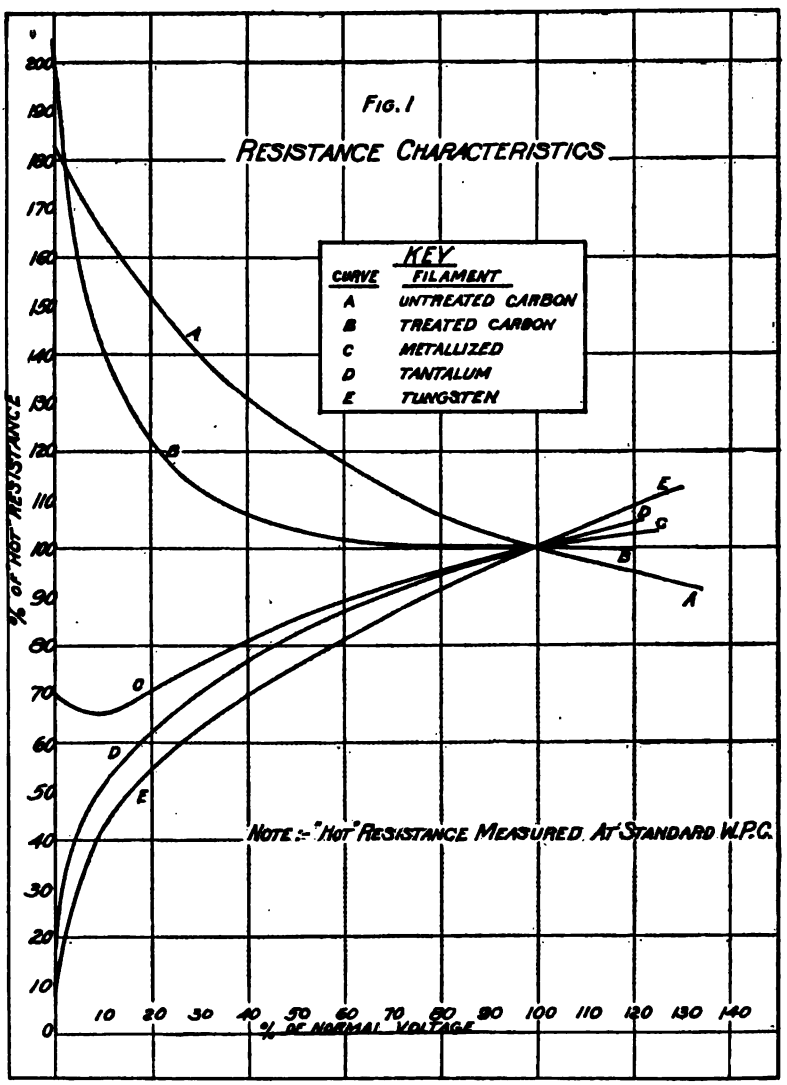
After many years of experimentation it was found possible to produce ductile tungsten wire, so that early in 1911 the drawn-wire lamp[13] was introduced. Tungsten lamps of both the squirted and drawn-wire filament types operated at from 1.13 to 1.31 w. p. c., depending upon the size.

It may be of interest to note that the early developments of metal filaments took place in Europe, the American manufacturers following. The later improvements, however, involving the continuous winding and the drawn wire, are distinctly American. Drawn wire was practically in an experimental stage in Europe at the time that many American manufacturers were making drawn-wire lamps exclusively.

In the above discussion reference has been made only to the filaments of incandescent lamps. With the exception of the method of mounting the metallic-filament lamps, the general manufacturing processes with regard to the glass work and exhaustion are about the same for all of the various types of lamps. Of course there are differences in the degree of exhaustion and there are many differences in the details of construction. It may be added that the great improvements in the art of manufacture of incandescent lamps are not entirely due to the developments of new filaments. For example, the moulded bulb is now used instead of a free, hand-blown bulb, reducing the variations in size to a very great extent. The use of the chemical exhaust instead of the mercurial pump is also a distinct advantage.[14]

Physical Characteristics of Incandescent Lamps.

Dimensions.—The dimensions of incandescent lamps vary considerably, depending upon the size and type. For a given size and



type, lamps of different voltages also vary. Table A includes typical figures:[15]

TABLE A. DIMENSIONS OF INCANDESCENT LAMPS.

Type	Watts	Volts	Diam. of Filament	Length of Filament
Tungsten	60	115	0.0020 in.	26.40 in.
Tantalum	50	115	0.00195	26.80
Metallized Carbon ..	60	115	0.0037	9.95
Carbon	60	115	0.0045	9.30
Untreated carbon ...	20	115	0.0028	4.57

In connection with the above values it should be noted that the dimensions of carbon-filament lamps vary with the amount of "flashing," and in the case of metallized-carbon lamps, the degree of metallization.

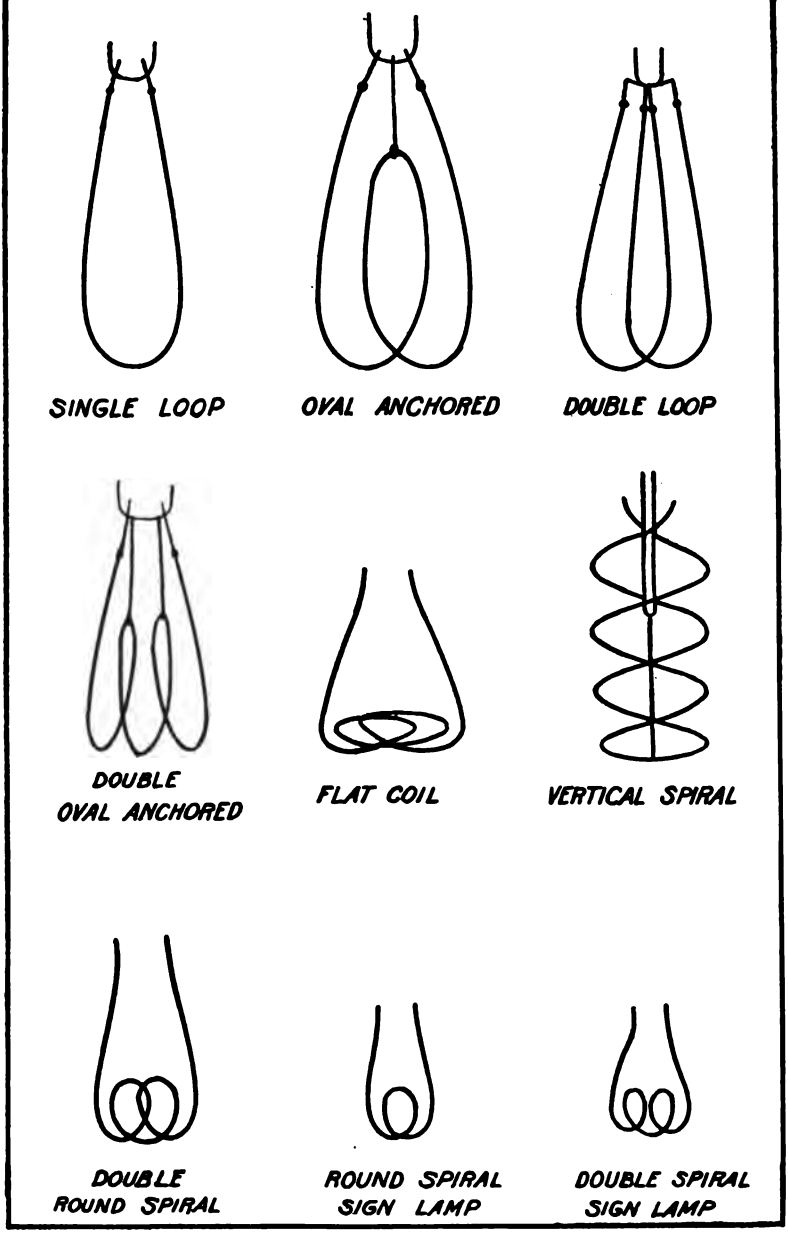
Temperature Coefficient.—Carbon-filament lamps, both treated and untreated, have negative temperature coefficients, the value for the treated filament at working ranges, however, being very small. The metallized filament, on the other hand, manifests increasing resistance with rise in temperature, in this characteristic resembling the metallic filaments. On Fig. 1 are shown curves of resistance change with change in voltage, the voltage at which the lamps operate at their standard watts-per-candle values being taken as 100 percent. In Table B are shown values of hot and cold resistance of particular samples of the principal types, the "hot" resistance being considered as the value at the working temperature (standard watts-per-candle).

TABLE B. RESISTANCES OF FILAMENT LAMPS.

Type of lamp.	Cold Resistance	Hot Resistance	Hot Resis. in Percent. of Cold Resistance.
20-W. untreated carbon	1062 ohms	582 ohms	55%
60-W. treated carbon	459	225	49
60-W. metallized filament.....	156	222	142
50-W. tantalum	44	262	595
60-W. tungsten	16.5	206	1250

Melting Point and Operating Temperature.—These temperatures have been determined by many experimenters, who have employed

FIG. 2 FILAMENT FORMS



several methods. Table C includes values which may be considered as approximate estimates:

TABLE C.

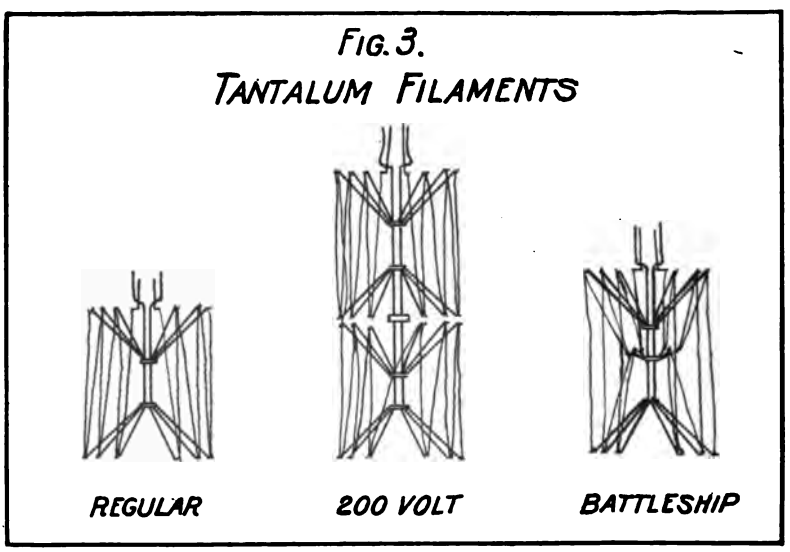
Type of Filament	Melting Point	Operating Temp.
Carbon	3750°C.	1800°C.
Tantalum	2800 to 2900	2100
Tungsten	2950 to 3100	2300

Specific Resistance.—Table D presents approximate values of specific resistance. In carbon lamps particularly, these values vary considerably among individual lamps:

TABLE D.

Filament.	Specific Resistance, Microhms per centimeter cube.
Untreated carbon	4200
Treated carbon	2000
Metallized carbon	450
Tantalum	13
Tungsten	6

Shape of Filament and Method of Mounting.—Carbon and metallized carbon lamps in sizes of 25 watts and larger are mounted

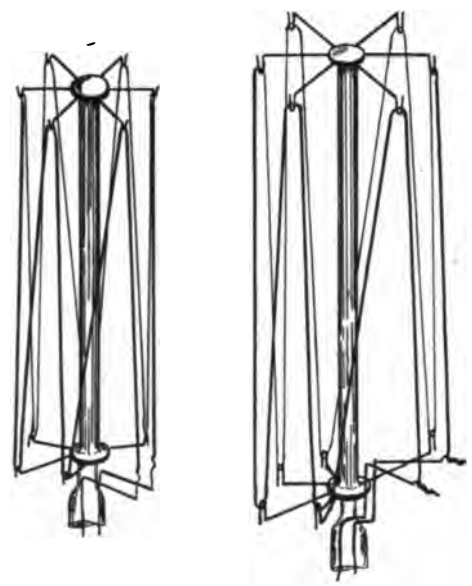


in the usual oval anchored form, the two ends of the filaments being clamped by means of a carbon paste to the platinum leading-in wires. The filament is supported at the middle by a nickel anchor wire. Carbon lamps of other voltages and of special forms may be mounted as single loop, double loop, spiral, vertical spiral,

gridiron shaped, etc. Sketches of these filaments are shown in Fig. 2. The three general forms for the mounting of tantalum lamps are illustrated in Fig. 3. The first shows a common style for lamps of 100-volt range, the second shows the type for lamps of the 200-volt range and the third illustrates the so-called battle-ship type. The mounting of tungsten filaments is illustrated in Fig. 4, which shows the usual mounts for the continuous filaments of the 100-volt range. Tungsten lamps of higher voltage have longer filaments and therefore more loops; for lamps of lower voltage the reverse is true.

FIG. 4.

TUNGSTEN FILAMENTS



DRAWN WIRE

SQUIRTED CONTINUOUS WOUND

In connection with the mounting of filaments it is of interest to note the losses due to conduction of heat by the leading-in wires. Certain typical lamps have been tested with the following results:[16]

TABLE E. LOSSES OF HEAT BY CONDUCTION.

Lamp.	Volts	W.P.C.	Energy Loss	Efficiency Loss	Dist. between successive contacts
Tantalum, 40-W ..	110	2.0	7%	13%	33 mm.
Carbon, 50-W.	115	3.1	2	4	106
Tungsten, 25-W ..	115	1.25	4	7	69
Tungsten, 60-W ..	110	1.25	4	7	80

It is observed that the greater the distance between the contacts, the smaller the losses. The efficiency loss as given in table E represents the difference between the average efficiency at which the filament burns and the maximum efficiency at the hottest part.

Specific Intensity (*Brightness*).—Recent experiments have shown the following values at standard operating efficiencies. For purposes of comparison the intensity of sunlight is added to this table:[17]

TABLE F.

Illuminant.	Specific Intensity, Candle power per sq. mm.
Tungsten	1.64
Metallized carbon	1.2
Tantalum	0.9
Carbon	0.75
Frosted tungsten (viewed from tip)	0.0026
Sun at zenith	900

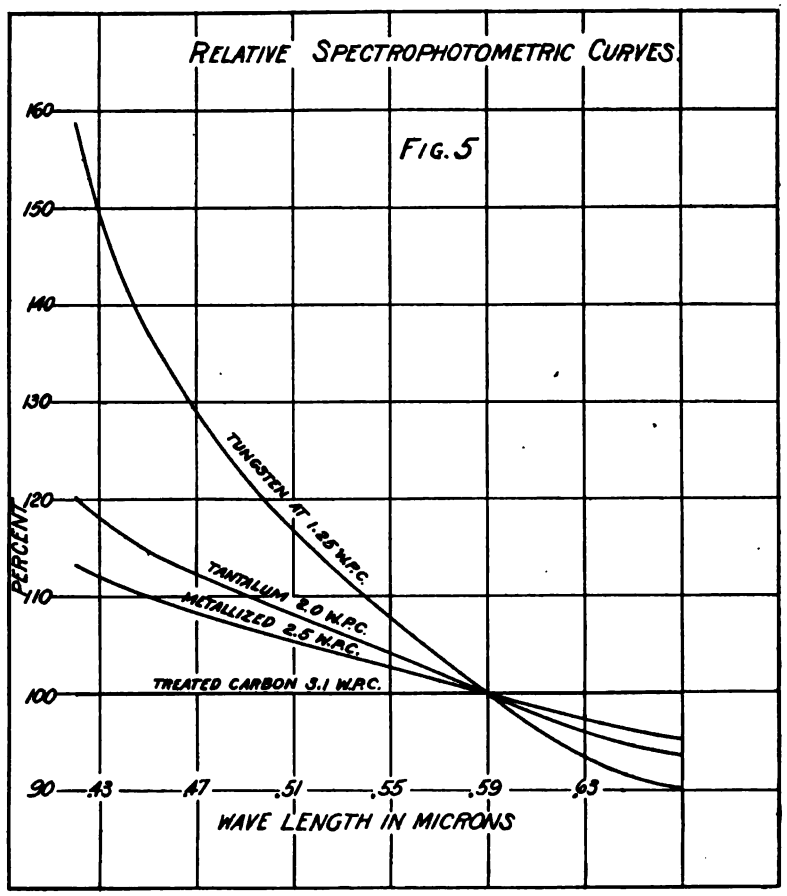
Color.—Experiments made with the Ives colorimeter yield the results shown in Table G.[18] These figures represent the percentages of standard red, green and blue required to be mixed in order to reproduce the color of the illuminant in question, considering the proportions for average daylight to be 100 percent. for each color. For convenience all results are reduced to a basis of 100 percent. in the red:

TABLE G.

	Red.	Green.	Blue.
Carbon at 3.1 w.p.c.	100	45	7.4
Metallized carbon at 2.5 w.p.c.	100	48	8.3
Tantalum at 2.0 w.p.c.	100	49	8.3
Tungsten at 1.25 w.p.c.	100	55	12.1
Average daylight	100	100	100

These figures show increasing preponderance of the blue and green from the carbon to the tungsten lamps, indicating that the tungsten lamp approaches more nearly to average daylight. This table also shows how far all incandescent filament lamps vary from

average daylight. Comparative spectro-photometric curves[19] are illustrated in Fig. 5, in which the radiations in the various wave lengths of the visible spectrum are shown for tungsten, tantalum, and metallized filament lamps as compared with carbon. For pur-



poses of comparison these curves have been brought together at 100 percent. at 0.59 μ in the yellow part of the spectrum.

Characteristic Curves.—Graphical representations of the variation of candle-power, watts, watts per candle, etc., with changing voltage are termed characteristic curves. Fig. 6 includes typical curves for the important types of lamps. It has been shown that over ordinary working ranges these curves may be considered as approximating very closely the parabolic form, and a series of ex-

ponents have been derived which may be applied to these curves with a reasonable degree of accuracy. The form of the equation in each case is:

$$\frac{X_1}{X_2} = \left(\frac{Y_1}{Y_2}\right)^K$$

in which Y is taken as voltage, and X as any one of the other variables.

In Table H are shown values of the various exponents for lamps of different types. These values are in more or less general use among lamp manufacturers, but should be considered as only approximate, particularly those referring to the relation between voltage and life.

TABLE H. EXPONENTS (K) IN ABOVE FORMULA.

	Untreated Carbon	Treated Carbon	Metallized	Tantalum	Tungsten
Candle-power	6.89	5.55	4.80	4.35	3.68
Watts	2.31	2.07	1.767	1.742	1.593
Watts per candle....	-4.59	-3.48	-3.06	-2.605	-2.102
Amperes	1.31	1.075	0.770	0.742	0.582
Life	-24.94	-20.26	-17.76	-16.22	-13.98

The exponents (b) for use in the equation:

$$\frac{\text{Life}_1}{\text{Life}_2} = \left(\frac{\text{W. p. c.}_1}{\text{W. p. c.}_2}\right)^b$$

as derived from the above, are given in Table I:

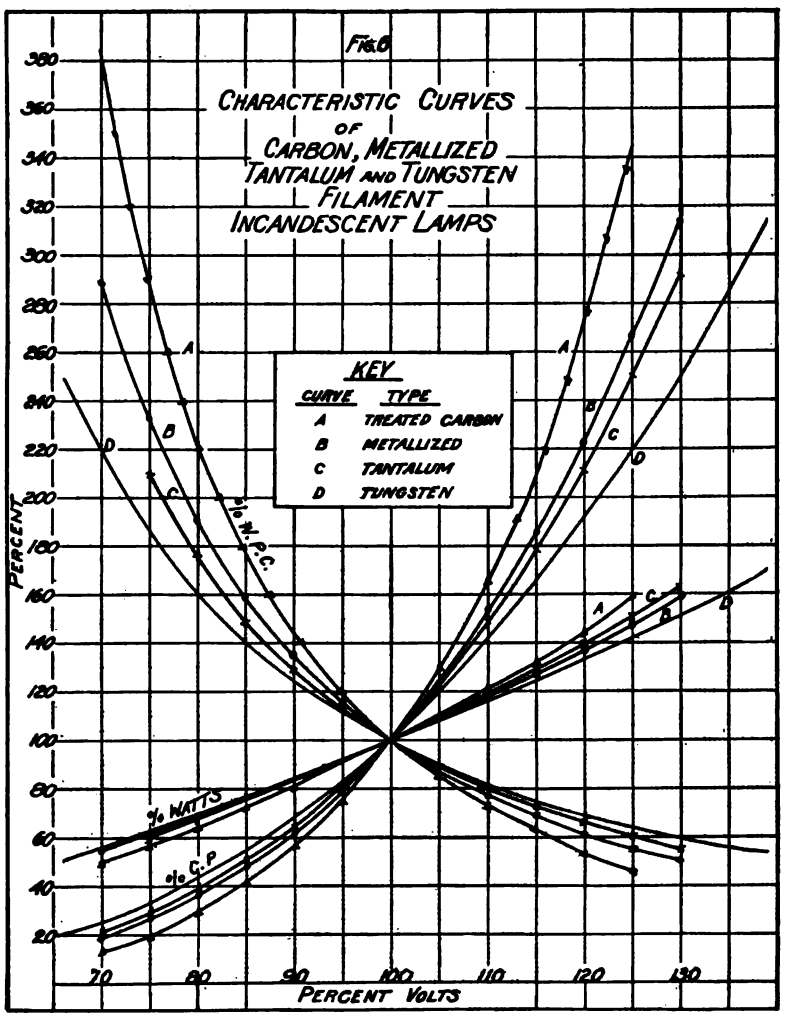
TABLE I. EXPONENTS (b) IN ABOVE FORMULA.

Tpye.	Exponent.
Tungsten	6.65
Tantalum	6.23
Metallized carbon	5.80
Treated carbon	5.83
Untreated carbon	5.43

As a matter of scientific interest, it should be remembered that the characteristic curves are really not true parabolic curves, and are more accurately represented by formulas much more complicated than the above [20] which, however, serve for all practical purposes.

Fragility.—Generally speaking, carbon filaments of the common

sizes are quite substantial and not liable to mechanical breakage unless handled very roughly. Metallized carbon filaments, candle-power for candle-power and size for size, show very little inferiority in this respect. Tantalum and drawn-wire tungsten filaments,



although requiring more careful handling than the older forms of metal filaments, are now constructed so as to obviate many of the early difficulties encountered in these types. The modern tungsten lamp with its continuous wire, loosely supported, is a great

improvement over the old type consisting of a number of loops fused together. Tests have demonstrated the superiority of the newer lamps with respect to ruggedness at every stage of their life.[21]

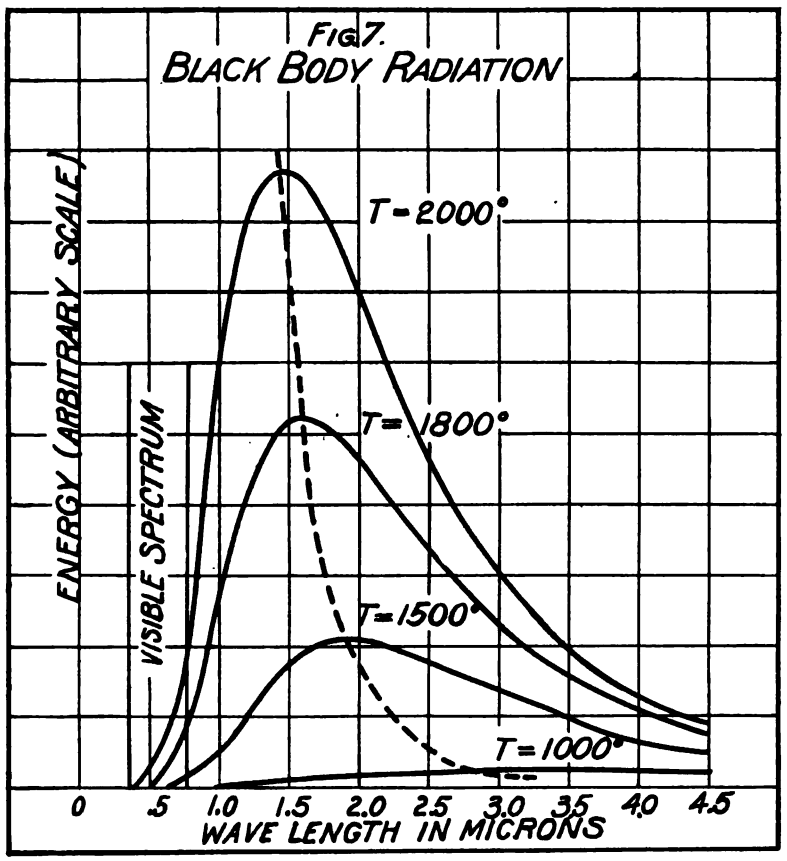
Efficiency, Life and Rating.

Efficiency.—The efficiency of an incandescent lamp is of extreme importance, not only as an expression of the economy of power consumption or of the usefulness of the lamp as an energy transformer, but also as having a direct bearing upon the life. Unlike most other expressions of efficiency, it is not a simple matter to express the ratio of energy output to energy input as a percentage. This is due to a number of considerations based upon some of the properties of light-giving bodies. A brief discussion of the most important of these points is in order.

A solid body, when heated to any temperature above absolute zero, gives off energy in the form of radiations of all wave lengths. The amount of energy is a maximum at some given wave length. As the temperature is increased, this maximum shifts towards the region of the shorter wave lengths and at the same time more and more of the energy is radiated in the visible spectrum, which ranges from approximately 0.4 to 0.8 microns (1 micron$=10^{-3}$mm). Thus the proportion of visible to total radiations, which has an important bearing upon the efficiency of a material for a lamp filament, increases with increase of temperature. At working ranges, the luminosity varies as the 12th power of the temperature.

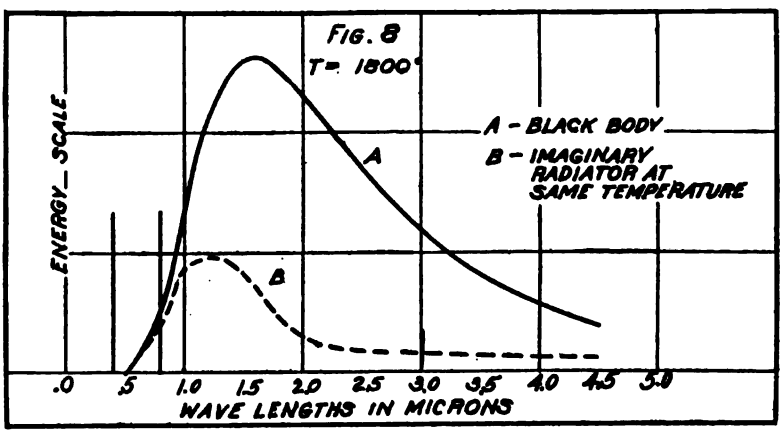
In order to study the radiations from solid bodies, a theoretical black body has been conceived.[22] Such a body is a perfect radiator and also absorbs energy of all wave lengths. An absolutely black body is of course impossible to attain, but a so-called experimental black body has been devised and studied. Fig. 7 illustrates the results obtained by Lummer and Pringsheim. A body which radiates less energy, of all wave lengths, than a black body at a given temperature, but in proportionately smaller amounts throughout the spectrum, is called a gray body. With a black or gray body, efficiency is gained purely by temperature increase. Selective radiation[23] is a term which is applied to a departure from black or gray body radiation, one possible form of which is illus-

trated in Fig. 8. Curve A shows the black body radiation at a given temperature, while curve B is the energy curve of an imaginary solid at the same temperature, showing comparatively small energy radiation in the invisible portion of the spectrum. Here it is quite evident that the proportion of visible to total radiation is comparatively high because of relatively greater reduction of



radiations in the invisible wave lengths, and that the temperature element is not the only one of importance as effecting the efficiency of an incandescent body. Metallic-filament lamps have been shown to exhibit to some extent the property of selective radiation, which accounts in part for their higher efficiency, although, as previously stated, most of the gain in efficiency is due to the possibility of higher operating temperature.

This leads to a discussion of luminous efficiency.[24] This term has been given at least two meanings, so that it might be sub-divided into (a) radiant efficiency — the ratio of visible radiations to total radiations; and (b) total efficiency — the ratio of visible radiation to energy input. The difference between these two efficiencies is considerable and varies in different types of lamps. It is represented by the losses by conduction through filament supports, leading-in wires, etc., and by convection of heat by currents in the surrounding medium. The former has been shown to be quite an important item, while under good conditions of operation the latter is almost negligible.[25] A great many experiments have been



made to determine radiant and total efficiency, and results of such investigations may be readily found in the technical press. Some very recent determinations show average values of about 1.6 per-cent. for treated carbon-filament lamps and 4.5 percent. for tungsten lamps. These values are stated as total efficiencies.[26]

Another measure of the efficiency of light production involves the determination of the mechanical equivalent of light, which has been discussed at length by numerous physicists and has been found to vary for different types of illuminants.[27] This equivalent is expressed as mean spherical candle-power per watt and has a value much larger than may be obtained in practice, because it is impossible to restrict the energy of transformation to the visible spectrum.

For ideal white light with the energy restricted to the visible radiations only, the mechanical equivalent would be in the neighborhood of 25 mean spherical candle-power per watt.

In any discussion of luminous efficiency certain characteristics of the human eye must be considered. Thus the luminosity curve of the average normal eye varies with the degree of illumination, the maximum luminosity being in the yellow portion of the spectrum for high intensities and nearer the green or shorter wave lengths at lower intensities.[28] Furthermore, a given amount of energy radiated in the red end gives the impression of very much less light than the same amount of energy radiated in the green. Another element of some importance is the proper limiting of the visible spectrum. Different experimenters have used varying values, the most common being from 0.36μ to 0.78μ, but often 0.40 μ, 0.70 μ and 0.80 μ are used to replace one or the other of the above. These various considerations, particularly the physiological effects which arise through the use of the human eye in measuring light, indicate that after all, for practical purposes, the measure of the efficiency of an incandescent lamp is the ratio of light produced, as measured by a photometer, to the power input, each of these terms expressed in suitable units.

The electrical power is of course measured in watts. The light intensity should be expressed in terms of mean spherical candle-power or in the unit of total flux, the lumen (equivalent to the mean spherical candle-power multiplied by 4π). Unfortunately, it has become the practice among manufacturers of lamps, and among many engineers, to use the inverse of this ratio, the watts per candle, and in many instances, the watts per mean horizontal candle-power is specified. This latter term is much to be deplored, because the ratio of mean spherical to mean horizontal candle-power is not by any means a constant for all types of lamps. Typical illustrations of these variations have been selected, although they do not by any means cover the entire field. Fig. 9 shows the distribution of light about two widely differing forms of carbon-filament lamps, the first being an oval filament and the second a vertical spiral. The values from which these curves are derived are reduced to a basis of equal mean spherical candle-power. Fig. 10 shows the distribution of light about a clear bulb and a bowl-

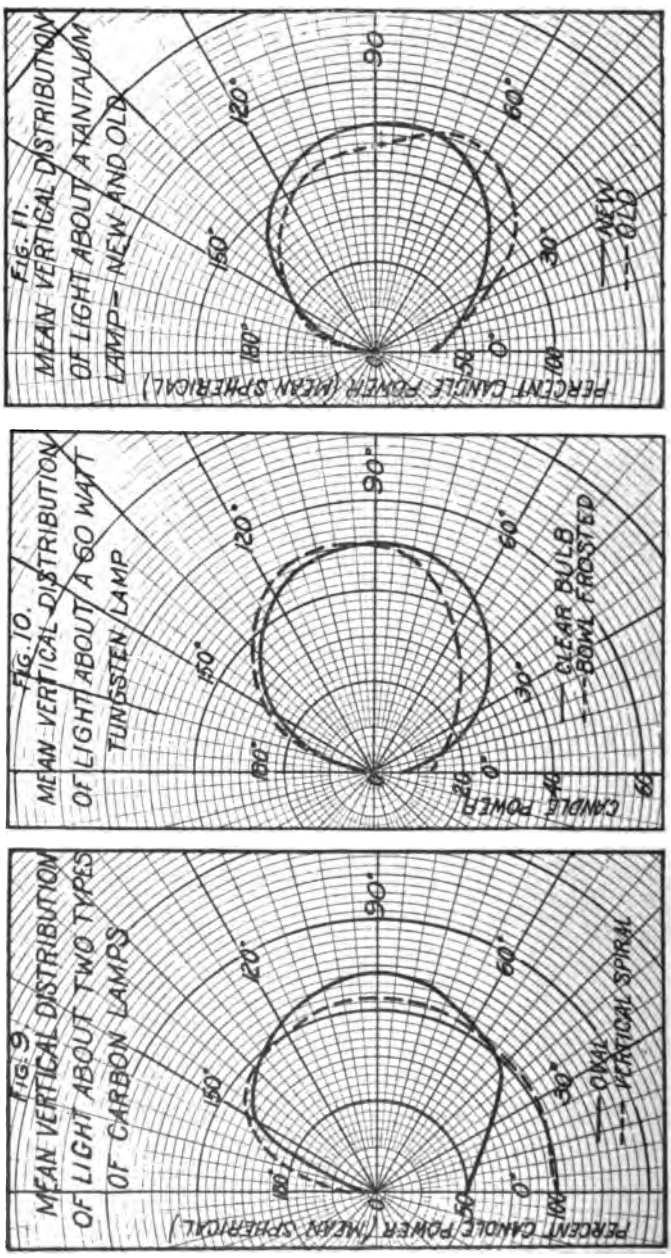


FIG. 9.
MEAN VERTICAL DISTRIBUTION OF LIGHT ABOUT TWO TYPES OF CARBON LAMPS
PERCENT CANDLE POWER (MEAN SPHERICAL)
90°
120°
150°
180°
60°
30°
0°
50
100
OVAL
VERTICAL SPIRAL
FIG. 10.
MEAN VERTICAL DISTRIBUTION OF LIGHT ABOUT A 60 WATT TUNGSTEN LAMP
CANDLE POWER
90°
120°
150°
180°
60°
30°
0°
60
30
0
CLEAR BULB
BOWL FROSTED
FIG. 11.
MEAN VERTICAL DISTRIBUTION OF LIGHT ABOUT A TANTALUM LAMP— NEW AND OLD
PERCENT CANDLE POWER (MEAN SPHERICAL)
90°
120°
150°
180°
60°
30°
0°
50
100
NEW
OLD

frosted bulb, 60-watt, tungsten lamp. These two curves represent, with a fair degree of accuracy, the curves of a given lamp before and after the frosting is applied. Fig. 11 shows the distribution of light about a tantalum lamp, new and after having burned for a time. (Spherical reduction factors, corresponding to these various types of lamps, will be taken up in the last section of this article.) These illustrations emphasize the necessity of mean spherical candle-power values, particularly in comparison of lamps of different types.

Life.—The life of an incandescent lamp, other things being equal, is dependent primarily upon the efficiency at which it is operated. Useful life is a term applied to the period during which a lamp of a given type may be operated economically. When incandescent lamps were first introduced, the total life, or life to failure, was alone considered. As early as 1887 there was some discussion as to the "smashing" point of an incandescent lamp.[29] The determination of this "smashing" point involves a number of factors among which are the cost of lamps, the rate of candle-power decrease, and the cost of current. The commercial "smashing" point involves not only the above considerations, but probably depends upon the appearance of the lamps after having burned for a time, the class of consumers using the lamps, the condition under which the lamps are burned, with particular reference to location. and the policy of the lighting company, especially if lamps are furnished upon a free renewal basis. Many determinations of the theoretical "smashing" point have been made, particularly in the case of carbon lamps. For testing purposes at least — and for commercial uses in many cases — the useful life has been stated as exhausted when the candle-power of a lamp has decreased to 80 percent. of initial value. This criterion has been applied to the metallized carbon-filament lamps and has in some cases been extended even to lamps having metal filaments. Unfortunately, in considering the useful life values of incandescent lamps, the life to 80 percent. or earlier failure has been used. This is really a hybrid value, including the period of a 20-percent. candle-power decrease for lamps which survive, and a period of smaller candle-power diminution for lamps which are early failures. Thus it becomes necessary to consider the values of which such a hybrid

figure is made up. The deleterious effect of a given proportion of very short life failures must be properly balanced against possible abnormally long life of some 80-percent. lamps, and to express results by means of a single figure other than total life, or life to complete failure, is apt to be misleading. Similarly, to burn lamps to total life is not recommended, because at the present stage of the art of manufacture many lamps burn for a considerable period after they have outlived their usefulness. It is hoped that a method of evaluation, which has already been brought before illuminating engineers, central station men, and lamp manufacturers, will be adopted in the near future.[80] In the meantime, statements of life to 80 percent. or burn out should always be accompanied by more detailed summaries showing the proportion of lamps which survive and the proportion of failures.

In addition to the efficiency at which the lamps are operated, the life of a given type is dependent upon certain important details of construction. For example, filaments which lack uniformity will develop spots of high resistance which will cause a rapid disintegration at the spot of high resistance, causing failure; filaments consisting of several independent loops joined together by a pasting or welding process, and therefore subject to variations in diameter, will also be likely to fail in a similar manner. Two lamps having different sizes of bulb, but the same type of filament, of the same wattage input, and operating at the same efficiency, may be expected to have different life values, the smaller bulb lamps having shorter life. The degree of vacuum is of course of prime importance; lamps that are not up to the standard, or that develop leaks, will perform poorly.

Rating.—The system of rating incandescent lamps has undergone a considerable change since the introduction of the metallized filament lamp. The treated carbon-filament lamps were for many years rated on a candle-power and watts-per-candle basis; lamps of 4, 8, 16, 32 c. p., etc., were common and were usually furnished at three watts per candle, namely, 3.1, 3.5 and 4 w. p. c., for use under different conditions. For example, 3.1 w. p. c. lamps were usually employed when very good voltage regulation was obtainable, while 4.0 w. p. c. lamps were used on poorly regulated circuits, particularly in railway work; they were also required in in-

accessible locations where long life was a very important consideration. It is to be understood that a 16-c. p. lamp consuming 4 w. p. c. lasts about four times as long as a 16-c. p. lamp operated at 3.1 w. p. c. This method of rating was quite unsatisfactory from the manufacturer's point of view because it necessitated a very large variety of lamps for stock purposes. This difficulty gave promise of increasing to enormous proportions when metallized, and later metal-filament lamps were introduced. About 1905 the wattage basis of rating, with its accompanying three-voltage labels was introduced, first for use in connection with the metallized-filament lamp. It was next applied to the tungsten and tantalum lamps and finally to the carbon lamps. According to this system, the labels bear three voltages at two-volt intervals, lamps of top voltage being used on circuits where the regulation is very good, lamps of middle voltage where regulation is fair, and bottom voltage on poorly regulated circuits. Of course certain economic considerations frequently enter into the choice of voltage. It must not be lost sight of that the three-voltage rating does not quite take the place of the three watts-per-candle ranges formerly employed. Under the new system the voltages are two volts apart, whereas there is a difference of over three volts between 3.1 and 3.5 w. p. c., and about four volts between 3.5 and 4.0 w. p. c. Moreover, under the old system lamps averaging 16 c. p., were available, consuming approximately 50, 56, or 64 watts; under the new system a 50-watt lamp consumes 50 watts and gives its rated candle-power only at the top rated voltage. If a bottom voltage lamp is required, it will be found to consume approximately 46 watts, and give correspondingly lower candle-power. According to the latest practice, lamps are no longer designated as top, middle or bottom voltage, but rather as high, medium and low efficiency.

Types of Incandescent Lamps for Various Uses.

Lamps are generally divided into two classes, known as multiple and series. These two general classes are further subdivided. Regular multiple lamps comprise lamps of the 100 to 130- or 200 to 250-volt ranges, the former being manufactured in the carbon, metallized carbon, tungsten, and tantalum types, while the higher voltage lamps are of untreated carbon, tantalum or tungsten. Multiple-burning sign lamps are usually of untreated carbon, although

it is hoped that drawn tungsten wire may be obtained sufficiently small in diameter to permit the manufacture of tungsten sign lamps for 100 to 130-volt circuits. Multiple-burning lamps of 30- and 60-volt ranges are also in considerable use for train lighting, these lamps being manufactured in the carbon, metallized carbon, tantalum and tungsten types. There is also a style known as compensator type, used in connection with transformers, on the usual 100 to 130-volt circuits, the lamps being of low voltage range. Multiple-burning lamps of voltages as low as four volts are obtainable. Many of the above styles are listed in round, tubular and special shaped bulbs as well as the usual straight-side, pear shaped.

Certain of the carbon and tantalum lamps regularly known as multiple burning are especially assorted at the factory with regard to current consumption rather than voltage, so that they may be burned five in series on 500 to 600 volts in street railway work. Metallized and tungsten-filament lamps have recently been designed for this purpose.

Series lamps are rated according to current consumption and may be obtained in ranges of 1.75, 3.0, 4.0, 5.5, 6.6, and 7.5 amperes, in various sizes from 32 to 350 c. p. These lamps are largely used in street lighting and are practically all of the tungsten type. Tungsten sign lamps of low voltage and low wattage for operation in series or multiple are available.

In addition to the above there are a large number of lamps for special purposes, such as telephone switchboard, surgical, miniature, decorative, etc. One of the recent developments is the automobile headlight which has a spiral filament arranged in conical form. Larger lamps of this general type are used for stereopticon work. Finally, lamps of high wattage and rather low efficiency are used for heating and medical purposes.

Lamp Performance and Data.

Performance Characteristics.—Before presenting incandescent lamp data it is well to consider the nature of their performance. It is found that practically all types of multiple lamps manifest an increase in candle-power during the early hours of life, after which there is a period of practically constant candle-power, followed by a gradual decrease in intensity. The amount of the

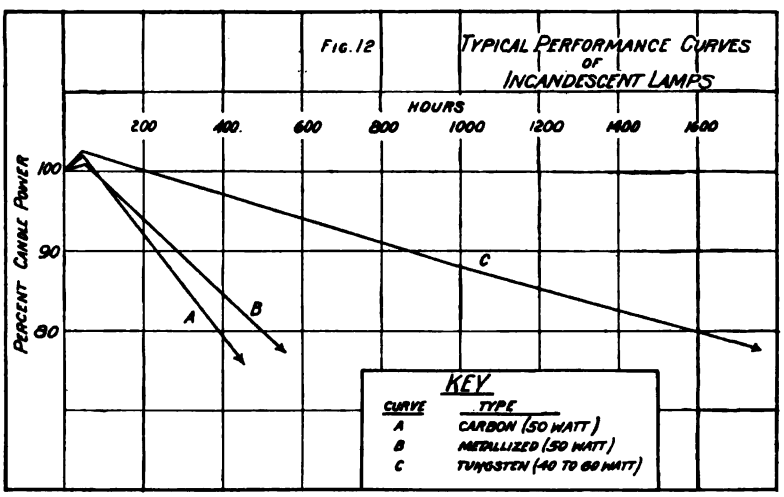
initial rise is dependent upon the degree of ageing of the lamp at the factory and upon the efficiency at which the lamp is operated. This rise in candle-power has been thought due to a knitting together of the filament material, incident to the expulsion of occluded gases and other impurities. The ageing referred to above is an operation designed to accomplish this very purpose, and the candle-power "peak," as it is called, is due to the fact that the ageing is not quite completed when lamps are put on the market. The candle-power increase is accomplished by an increase in efficiency, and it is apparent that improperly aged lamps would be subjected to the danger of a rapid rise in efficiency during the first few hours, accompanied by increased temperature, and therefore rapid disintegration of the filament with consequent early failure. Tantalum lamps usually show the greatest candle-power peak, and the metallized carbon, the smallest. The candle-power increase during the early hours is accompanied by an increase in wattage in the case of carbon, tantalum and squirted-filament tungsten lamps only. The metallized carbon and drawn-wire tungsten lamps, as put on the market at present, begin to fall off in wattage as soon as they are placed in circuit. It may be said that in making incandescent lamp tests it is not usual to make an exhaustive research into the early performance of the lamps except in special cases. It is usually sufficient to fix some arbitrary number of hours predetermined by test and dependent upon test efficiency, and to make what is called the "peak measurement" at this time.

Lamps which are not of sufficiently high vacuum decrease rapidly in candle-power, and if air leaks into the bulb rapidly, will burn out within a short time after being placed in circuit. Some lamps, however, which initially exhibit some of the characteristics of poor vacuum because of the presence of certain gases in the bulb may during the first few hours of burning decrease rapidly in candle-power, this decrease being accompanied by an improvement in the condition of the vacuum after which the rate of candle-power drop becomes almost normal. Such lamps may have long life, but are not desirable because of their poor candle-power maintenance.

Fig. 12 shows typical performance curves of incandescent lamps which may be said to perform normally, and do not fail early.[86]

These curves should not necessarily be considered as averages, for they will vary considerably, even among individual lamps of any one make.

Bulb Discoloration.—As incandescent lamps grow old in service, the candle-power decreases, partly through decrease in wattage due to the increasing resistance of the filament caused by disintegration and partly through the discoloration on the inside of the bulb which is usually a deposit of the filament material. When carbon or metallized-filament lamps are defective, due to poor exhaust, the discoloration is apt to be brownish and mottled. In some metallic-filament lamps part of the discoloration may be caused by a deposit of the support material. When poor vacuum develops during the



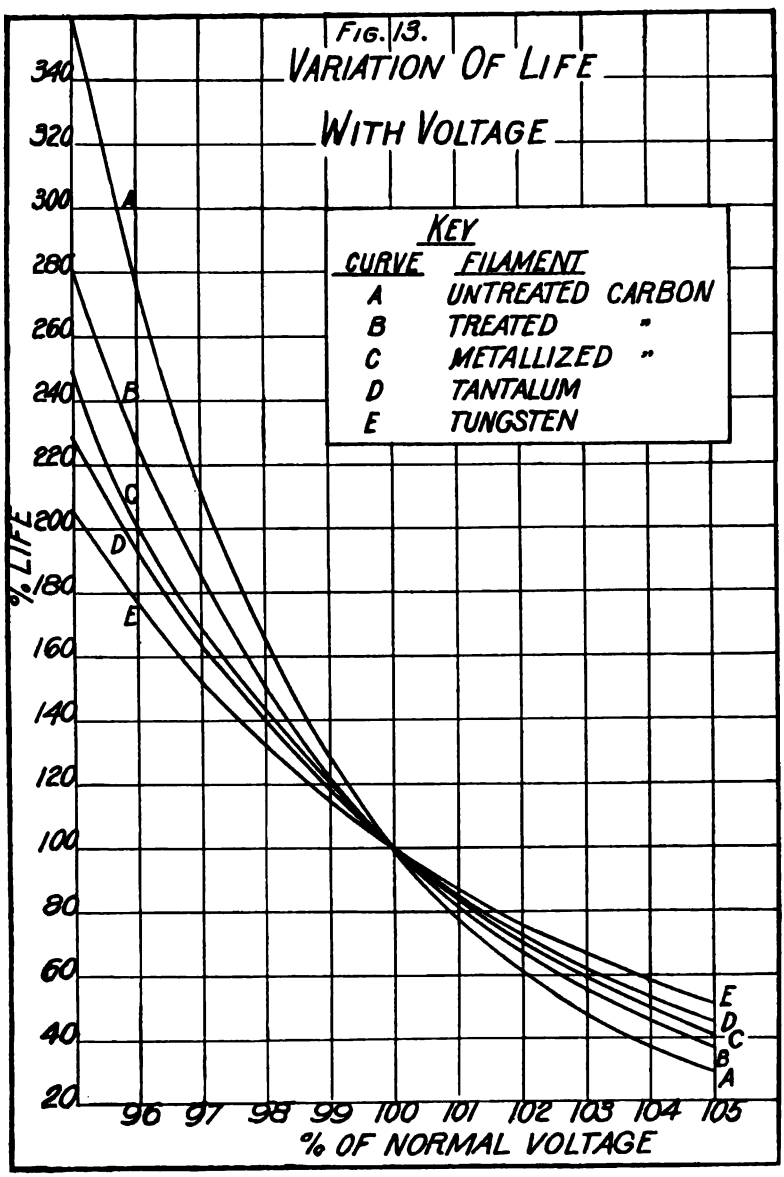
life, particularly in the case of metallic-filament lamps, the bulb often blackens very rapidly and to a great degree. In tantalum lamps the discoloration forms a band about the bulb in the region of the filament only, the lower part of the bulb, as well as the upper part, being quite free from blackening. This increases throughout life, causing a marked change in the ratio of mean spherical to mean horizontal candle-power. There is evidence that this same effect is found in drawn-wire tungsten lamps, although to a considerably less extent. Investigations made in connection with carbon lamps have shown that the decrease in candle-power throughout the life to 80 percent. is about equally distributed between the two causes which produce it; namely, wattage decrease and bulb

discoloration.[31] It is reasonable to expect that metallized-filament lamps will show less discoloration at 80 percent. of initial candle-power because of the more rapid decrease in wattage.

Frosted Lamps.—Incandescent lamps are frequently frosted, either wholly or in part, for decorative purposes or in order to decrease the specific intensity. The effect of frosting, with particular reference to carbon lamps, has been investigated and it has been found that the absorption coefficient of the frosting on lamps of 50-watt size varies from 4 to 7 percent. (on a basis of mean spherical candle-power).[32] This refers to totally frosted lamps. There has been no evidence to indicate that the total life of frosted lamps differs from that of clear bulb lamps of the same types whose filaments are operating at the same efficiency. The rate of candle-power decrease, however, is considerably higher in frosted lamps, thus shortening the life to 80 per cent. of initial candle-power. Experiments have shown that the 80 percent. life of frosted lamps is about one-half that of the clear lamp operating at the same efficiency. At the end of the useful life, at which point the candle-power is 80 percent. of the initial value, the diminution in candle-power has been effected as follows: Increase in resistance in filament reduces candle-power to 97 percent. Bulb discoloration, as in a clear bulb lamp, decreases this value to 92 percent. Additional absorption by the carbon deposit, to which the frosting subjects a certain portion of the life, further reduces the candle-power to 87 percent., while dirt on the surface of the bulb finally reduces the candle-power to 80 per cent.[33]

Variation of Life with Voltage.—As indicated under the second topic of this article, an increase in voltage above the normal causes a decrease in life. Roughly, 1 percent. increase in voltage causes a decrease of 22 percent. in the life of an untreated carbon-filament lamp, 18 percent. for the treated carbon, 16 percent. for the metallized-filament lamp, 15 percent. for the tantalum lamp and 13 percent. for the tungsten lamp. Fig. 13 shows approximately the variation in life with change in voltage, the curves being based on the exponential equations given in Table H. It is of course understood that these curves apply in the case of slowly changing voltages only. If the voltages are changing rapidly, as for example the cyclic changes in an alternating current, the thermal capacity

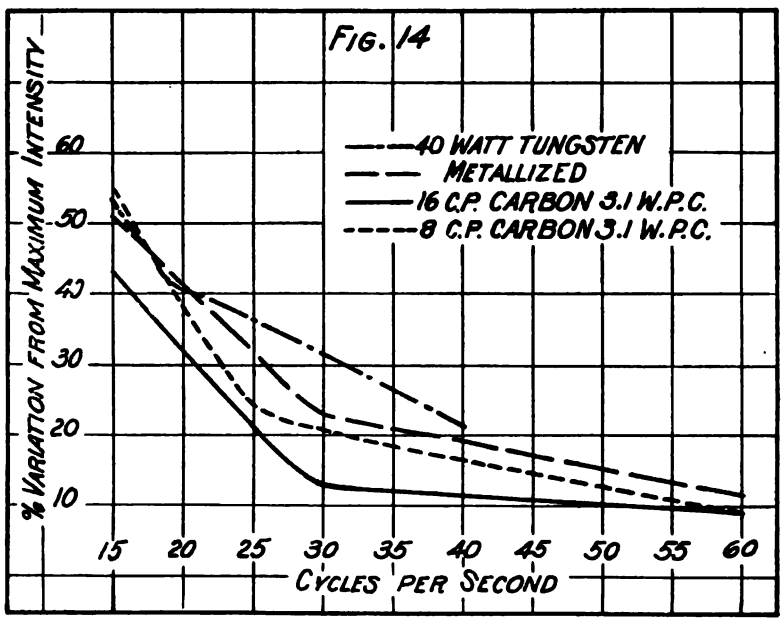
of the filament equalizes to a great extent the fluctuations in the filament temperature.



Effects of Frequency.—The effect of frequency on the life of incandescent lamps is practically negligible except in the case of

the tantalum lamp. This filament, as the number of hours of burning increase, manifests a certain off-setting or breaking-up which is found to some extent in lamps operated on direct current circuits, and to a very great extent on alternating currents. The off-setting increases with the frequency. The life on alternating current circuits (60 cycles) is only about one-half of the life on direct current at the same efficiency.

The effect of frequency on the candle-power of lamps of all types is very important. If the frequencies are too low maxima



and minima of intensity are manifested, producing an unpleasant flicker. As the frequencies increase, the flicker becomes less perceptible due to the persistence of the retina, as well as the thermal capacity of the filament. For a given type of lamp, therefore, lamps of larger sizes are less likely to flicker on a given frequency. The results of stroboscopic tests showing sensitiveness of certain typical lamps to frequency are shown in Fig. 14. The indication is that at 30 cycles only the higher candle-power, carbon-filament lamps are satisfactory, while metal-filament lamps are satsifactory at 40 cycles.[34] At 60 cycles the flicker has practically disappeared for all types.

Lamp Data.—In Table J is shown the rating of the more important types of incandescent lamps in common use at the present time. These values are manufacturers' data and are in terms of mean horizontal candle-power. Mean spherical candle-power values may be determined by applying the proper spherical reduction factors. All ratings are given at top voltage only.

TABLE J. RATING OF INCANDESCENT LAMPS.

Multiple Burning — Top Voltage Only.

Size	Mean Horiz. Candle-power	Watts	Watts Per Mean Horiz. C.P.
100 TO 130-VOLT RANGE:			
Untreated Carbon (Sign Lamps)			
10 watts	2.0	10.0	5.0
20	4.8	20.0	4.15
Treated Carbon			
20	4.8	20	4.15
25	8.1	25	3.10
30	9.3	30	3.23
50	16.8	50	2.97
60	20.2	60	2.97
100	33.7	100	2.97
120	40.4	120	2.97
Metallized Carbon.			
30	10.0	30	3.00
40	15.6	40	2.56
50	20.0	50	2.50
60	24.0	60	2.50
80	32.5	80	2.46
100	40.7	100	2.46
Tantalum — Direct Current.			
25	12.7	25	1.97
40	22.3	40	1.79
50	27.9	50	1.79
80	44.7	80	1.79
Tungsten.			
25	19.1	25	1.31
40	32.5	40	1.23
60	50.8	60	1.18
100	84.7	100	1.18
150	127	150	1.18
250	221	250	1.13
400	354	400	1.13
500	442	500	1.13

Size	Mean Horiz. Candle-power	Watts	Watts Per Mean Horiz. C.P.
200 TO 250-VOLT RANGE:			
Untreated Carbon.			
35	7.95	35	4.40
60	16.26	60	3.69
120	32.52	120	3.69
Tantalum — Direct Current.			
50	25.4	50	1.97
80	40.6	80	1.97
Tungsten.			
40	30.5	40	1.31
60	45.8	60	1.31
100	76.4	100	1.31
150	115	150	1.31
250	191	250	1.31
500	407	500	1.23

Table K shows middle and bottom voltage watts-per-candle corresponding to the top-voltage values in Table J. These figures are of course approximate.

TABLE K. WATTS-PER-CANDLE VALUES AT TOP, MIDDLE AND BOTTOM VOLTAGES.

Type.	Top Voltage W.P.C.	Middle Voltage W.P.C.	Bottom Voltage W.P.C.
Treated carbon	3.23	3.46	3.69
	3.10	3.31	3.52
	2.97	3.18	3.39
Metallized	3.00	3.18	3.36
	2.56	2.71	2.89
	2.50	2.65	2.81
	2.46	2.60	2.78
Tantalum	1.97	2.05	2.14
	1.79	1.87	1.95
Tungsten	1.31	1.37	1.43
	1.23	1.28	1.33
	1.18	1.23	1.28
	1.13	1.18	1.23

Table L shows life values (total hours of burning) at top voltage only. Values at middle and bottom voltage may be estimated by means of the curves given in Fig. 13. The figures shown in the table are manufacturers' data and are stated as conservative estimates of the total life in average commercial practice.

TABLE L. TOTAL LIFE VALUES (HOURS) AT TOP VOLTAGE.

Type.	Untreated Carbon	Treated Carbon	Metallized	Tantalum	Tungsten
		100 TO 130-VOLT RANGE.			
10 watts	2000	....	....	....	....
20	1500	1500	....	....	....
25	....	500	....	1000	1000
30	....	1050	1050	....	....
40	....	....	600	800	1000
50	....	700	700	800	....
60	....	700	700	....	1000
80	....	....	700	600	....
100	....	600	650	....	1000
120	....	600	....	....	....
150	....	....	....	....	1000
250	....	....	....	....	1000
400	....	....	....	....	1000
500	....	....	....	....	1000
		200 TO 250-VOLT RANGE.			
35	1000	....*	....	....	....
40	....	....	....	....	1000
50	....	....	....	800	....
60	750	....	....	....	1000
80	....	....	....	800	....
100	....	....	....	....	1000
120	750	....	....	....	....
150	....	....	....	....	1000
250	....	....	....	....	1000
500	....	....	....	....	1000

* Special treated-filament 200 to 250-volt lamps are made in sizes from 55 to 100 watts, their life values at 3.35 to 3.05 w.p.c. varying from 700 to 500 hours.

Table M contains rating and life values of miscellaneous types. The data have been drawn from the same source as those in Tables J, K and L.

TABLE M. RATING AND LIFE OF MISCELLANEOUS TYPES.

Type	Watts	W.P.C.	Total Life
	100 TO 130-VOLT RANGE.		
Street Railway—carbon	42	4.1	2000 hr.
"	64	4.0	2000
"	114	3.5	1500
tantalum	37	2.3	1700
	LOW VOLTAGE		
Train lighting — tungsten	15 to 50	1.23	1000 hr.
Sign — tungsten	2.5 to 5	1.33	2000
"	10	1.23	800
Series — tungsten	38 to 413	1.18	1350

Spherical Reduction Factor.—The ratios of mean spherical to mean horizontal candle-power of incandescent lamps of various types of filament are shown in Table N.

TABLE N.

Type.	Spherical Reduction Factor
Carbon filament:	
Oval anchored	0.825
Single loop	0.810
Double loop	0.805
Spiral coil	0.880
Flat elliptical coil	0.985
Vertical spiral	0.940
Metallized:	
Oval anchored	0.825
Tantalum	0.77
Tungsten:	
Squirted	0.78 to 0.79
Drawn	0.76 to 0.78

These widely varying figures once more point out the importance of considering only mean spherical candle-power in the intercomparison of lamps of different types. In the case of lamps such as tantalum, and possibly the drawn-wire tungsten, where the spherical reduction factor changes throughout the life,[35] a mean, horizontal, candle-power basis would be practically intolerable, and if used at all, should always be properly qualified.

Conclusion.—In the foregoing, due to the limitations of space, it has been necessary for the author to omit many interesting details. It is hoped that the following list of references which, for the most part, represent records of researches by eminent authorities, will be of some use to the reader as sources of further information on the subject.

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PUMPING BY COMPRESSED AIR.*

BY HERBERT T. ABRAMS.†

The field in which compressed air can be used for lifting and forcing water may be divided into four distinct methods:

1. Pumps of the ordinary direct-acting type driven by compressed air;
2. Pneumatic displacement pumps;
3. The "return-air" system:
4. The "air-lift" system.

Direct Acting Pumps.

We will pass briefly over the use of air in place of steam for operating ordinary mechanical pumps. High economy must not be expected in a combination of this kind, for the reason that pumps of this type, of themselves, are wasteful of power, and generally speaking, no consideration is given as to whether they are to be run by steam or by air, no attention is paid to proper ratio of cylinders, and seldom is there any attempt to re-heat the air either before using it in a simplex or duplex pump, or between stages in a compound pump.

Compressed air should be used in a great many instances where steam is now being wasted, but we are so inclined to listen to talk about "compressed air efficiencies" meaning always "compressed air losses," that we frequently lose sight of the more important final analysis, which is to define whether a saving in fuel is an actual credit or whether such apparent balance is not over-balanced in the item of extra labor or up-keep or general utility.

As compressed air will be found driving steam pumps in mines, quarries, and wherever there is an available supply of air, it may be interesting to glance at the simple rules used for proportioning pumps and the volume and pressure of air needed to run them.

The simplest way to find the quantity of free air and the pressure required is first to determine at what speed and under what air pressure the pump is to be run, and calculate the cubic feet displaced

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per minute at the pressure necessary to do the work. This may readily be converted into terms of free air at sea level, or the greater quantity at any altitude. To this must be added a percentage, say 10 per cent. for loss due to leakage.

With a given air pressure and head of water, the diameter of the air cylinder may be found by the rule: Determine the pounds pressure of resistance by taking one-half the elevation in feet, as representing the equivalent pressure in pounds, add one-third excess for pumps over 7-in. stroke, and one-half excess for those under 7-in. stroke. Divide this result by the pounds pressure of air at the pump, and the result will be the proper ratio.

Knowing the dimensions of the air cylinder and pressure, we can calculate the free air requirements by means of the simple formula,

$$V = 1.1 \frac{A.\ S.\ P.}{144\ P_a}, \text{ where:}$$

V = Volume of free air in cubic feet per minute,

S = Piston speed of pump in feet per minute,

P = Absolute air pressure at pump,

A = Area of air cylinder,

P_a = Atmospheric pressure at any elevation.

Problem.— Pump assumed over 7-in. stroke; head, 360 ft.; air pressure, 60 lb.

To find area of air cylinder add one-third to the head (360 plus $\frac{1}{3} = 480$ ft.). Take one-half of this head as the equivalent pressure (240 lb.). Dividing 240 lb. by 60 lb. pressure, gives us a ratio of 4, or in other words, the air cylinder should have four times the area of the water cylinder.

Having decided on a pump to handle 50 gal. of water per minute at a piston speed of 100 ft. as the proper size, we find, either by calculation or by manufacturers' tables, that the water end is represented by $3\frac{1}{2} \times 7$-in. cylinder. Our ratio having been established as 4 to 1, the diameter of the air cylinder will be 7 in. and its area 38.5 sq. inches.

Now, substituting in our formula the values so far obtained, we have,

$$V = 1.1 \frac{38.5 \times 100 \times 75}{144 \times 15} = 148 \text{ cu. ft. free air per min.,}$$

showing a total efficiency referred to I.H.P. in steam cylinder of compressor of approximately 19 per cent. If the air is reheated to 300°, the requirements will be reduced about one-third.

Pneumatic Displacement Pumps.

Fig. 1 is a sectional view of a compressed-air displacement pump. This pump uses air non-expansively, but owing to the fact that air comes in direct contact with the water pumped, without the intervention of moving parts such as pistons or plungers, the work under moderate heads, up to 100 ft., is done in a satisfactory and efficient way.

The pump consists of one or two cylinders with inlet and discharge valves for water, and a valve mechanism controlled by floats or other means for admitting compressed air directly to the surface of the water within the tanks, and for exhausting the air after the water has been expelled. The tanks must be submerged in a suitable sump or cistern properly protected, of sufficient depth to insure that the pump tanks will fill properly by gravity. It will be seen that the pump is really a simple apparatus. Water enters through the large inlet valves at the bottom. Air pressure entering at the top is alternately delivered to the chambers, expelling the water in them through the discharge valves and pipe. When a tank is nearly empty, the contained air is exhausted through a port in the valve mechanism, while water replaces it through the foot valve as soon as the air pressure is relieved, the alternate discharge being entirely automatic.

The proper field of this device is that calling for the pumping of water at some distance from the source of power, where there is an unobstructed supply and a limited lift. Valves in the power house on the air line to the pump, or to as many of them as may be needed to pump from a number of sources, control the operation, and pumping must continue as long as air is supplied. The advantages are that these pumps can be easily installed, and where an existing supply of compressed air is available, the cost of the equipment is low. There is no pump house to build and maintain at the water supply; the operation is controlled in the engine room; there are no pistons or plungers to repair and renew, and the wear on valves is less than in any ordinary direct-acting mechanical pump. Neither lubrication nor packing is required and the attention charges are reduced

to the minimum. Its working is not affected by muddy or gritty water. These facts adapt the pump for dye works, bleacheries, pulp mills, and for handling solutions. It may also be used to take

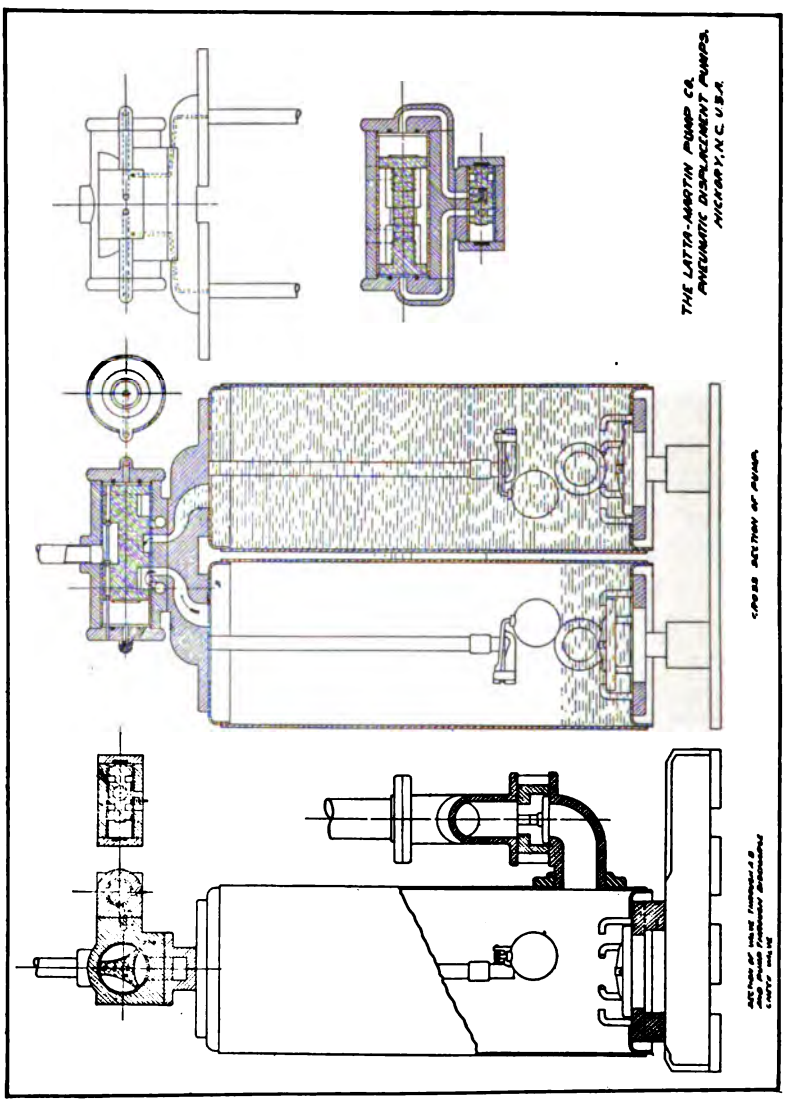

FIG. 1. PNEUMATIC DISPLACEMENT PUMP.

water from a cistern or reservoir filled by the air-lift from wells, or placed in a remote sump in a mine or quarry, it can be employed to deliver water to the main pumps.

Since the pump operates by the direct displacement of a volume of liquid by a volume of air under suitable pressure, it follows that simple rules can be given for arriving at the volume of air and pressure needed to do the work:

Reduce the quantity of water in gallons per minute to cubic feet. Multiply the total head (static head plus water pipe friction) by 0.434; this gives the working air pressure at the pump. Next find the ratio of air pressure to atmosphere which, multiplied by the cubic feet of water per minute, gives the cubic feet of free air per minute needed at the pump to displace this volume of water. Knowing the volume of free air required, 5 to 10 per cent. should be added for leakage and clearance according to conditions. This is net, actual, free air and not piston displacement of compressor; therefore, in figuring on a compressor to do the work, volumetric efficiency must be taken into consideration, following the general practice in such cases, according to the pressure, size and type.

Problem.— Source of supply, a river; distance from power house, 1000 ft.; lift, 60 ft.; diameter discharge main, 4 in.; quantity of water required per min., 150 gal.

150 gal. $= 150 \div 7.5 = 20$ cu. ft.

Static head.....60 ft.
Friction head 150 gal. per min. in 4-in. pipe 1000 ft. long..16 ft.

Total head.....76 ft.

$$76 \times 0.434 = 33 \text{ lb.}$$

$$\text{Ratio } r = \frac{33 + 15}{15} = 3.2$$

Free air $= 20 \times 3.2 = 64$ cu. ft. per min.

Allow 10 percent. for leakage and clearance $= 70$ cu. ft. net.

If the compressor selected will show a volumetric efficiency of 80 percent., the piston displacement required will be $70 \div 0.80 = 88$ cu. ft. per. min.

$$\text{Theoretical horse-power} = \frac{150 \times 8.33 \times 76}{33{,}000} = 2.88$$

I. H. P in steam cylinder of compressor $= 8.5$

Total efficiency $= 2.88 \div 8.5 = 33$ percent.

Return Air System.

The elevation or transfer of water and other fluids or semi-fluids by direct displacement with compressed air is so natural and self-evident a proposition as to need almost no suggesting. It appeals at once merely on the grounds of convenience and simplicity. But the principle as ordinarily applied has been open to the objection that it was not economical.

One characteristic of the ordinary, plain, displacement pump is waste of power entailed by direct release of the displacing air after the fluid is ejected from the pump tank. This air, after doing its work, is still at practically full pressure, therefore having all its potential energy of expansion. Its direct exhaust into atmosphere after displacement is the throwing away of this expansive power without any useful effect.

If air pipes are carried back from the pump to the compressor so as to add the residual pressure after displacement to the reverse side of the compressor piston, where it will help in compressing air into the pump tank, we have a return-air, expansive displacement pump, operating at good economy.

It may be well to emphasize, at this point, the fact that both the plain displacement pump and the "return-air" system have the following advantages, which are unique in that, while other pumping systems have some of them, no other systems combine them all to such remarkable degree:

1. Simple in construction, installation and management.
2. Wholly automatic in operation.
3. Under instant and complete control from the compressor room, however distant.
4. No stuffing boxes, glands, packing, pistons, plungers, in contact with the fluid.
5. No delicate mechanism exposed to corrosive or abrasive action.
6. No oil or other lubricant required by the pump.
7. Not affected by any excess submergence except that operation is improved under a hydraulic head.
8. Applicable to the pumping of all fluids or semi-fluids, as well as mud, sand, or any débris which will pass the valves.

Where water is to be pumped, the field of the pneumatic displacement pump and the return-air system is that in which large

volumes are to be handled from a generous source of supply, where complete submergence can be secured, such as in a river, lake, mine or quarry sump, large spring or an excavated well. This, at once, opens to them the supplying of water for small municipalities, shops, mills, factories or country estates; the pumping of condensing water for power plants of any kind; supplying railway water tanks, ice factories and cold storage warehouses, packing houses and breweries; supplying stock farms, irrigation systems and sugar factories; station pumping in mines and drainage pumping from mine or quarry sump.

The fact that these systems will handle anything sufficiently fluid to pass their valves adapts them also to the handling of solutions in salt works and bleacheries; the moving of semi-fluids, as in pulp mills; the handling of cement, slurry and marl; the pumping of glass sand for glass factories; elevating salt solutions in salt works; and the pumping of sewage.

Under normal conditions it is safe to say that the return-air system will show an average efficiency of 50 to 55 percent., this efficiency being the ratio of horse-power of water lifted to the indicated horse-power in the steam cylinder of the air compressor, including all losses. It is easy to estimate from these figures just what they mean in terms of fuel saving. Yet the question of fuel economy is but one argument in favor of the return-air system. Another argument is that the compressed air system may be operated with a far less boiler horse-power, with a corresponding reduction in first cost of boiler plant, interest, maintenance, repairs and firing cost. Where a plant is crippled by a shortage of boilers, the substitution of the return-air system for a less economical pumping system will relieve the embarrassment.

Continuing still further the argument for this system, it is to be noted that the tank equipment needs absolutely no care. The operation being entirely under control from the engine room, the engineer can at any time start, stop or vary the pumping. There is no possibility of drowning out the system. In fact, it operates with proportionately less power as the height or head of water above the tanks increases. It cannot become choked by such silt or dirt as may enter a mine sump.

The return-air system cannot be classed as a pump, or as ma-

chinery of the kind regularly catalogued and listed. It is a complete system in itself, consisting of an air compressor, a receiver, a reversing switch, two air lines, each leading from the compressor through the switch to one pump tank, and two pump tanks, all in a closed circuit, in which, by a regular cycle of operations, one

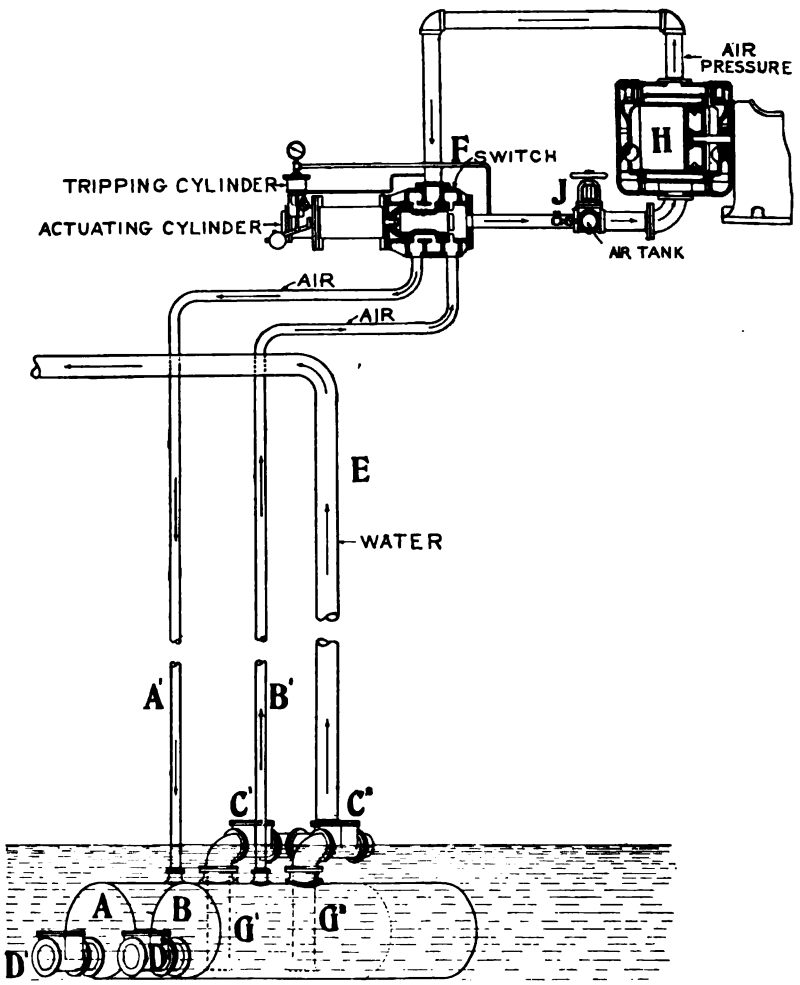

FIG. 2. RETURN-AIR DISPLACEMENT SYSTEM.

air volume is compressed, expanded, re-compressed and re-expanded, corresponding with a discharge of one of the pump tanks.

The principle of operation is simple. Compressed air is admitted to a tank filled with water, or any fluid, forcing the fluid

out through a check valve and pipe line, and at the same time the compressor is drawing air from the other tank, the charge of air being regulated so that when a tank is empty, the other is full, at which time the switch reverses, thereby reversing the action of the tanks.

The cycle of operation will be better understood by referring to Fig. 2: A and B are twin displacement tanks, preferably completely submerged, though they operate if so placed as to be filled by siphon action. A′ and B′ are the two air pipe lines. C^1 and C^2 are discharge check valves preventing the return of the fluid ejected.

D^1 and D^2 are check valves preventing the discharge of the fluid through the inlet. E is the discharge pipe from both tanks.

F is the automatic switch controlling the pumping cycle. G^1 and G^2 are the tank risers.

H is the compressing cylinder of an air compressor. J is the automatic compensating valve which keeps the system supplied with air.

In the diagram as shown, air is being withdrawn by the compressor H from the tank B and is being compressed in tank A. At the same time the fluid is entering the tank B through check valve D^2, while it is being forced from the tank A through the riser G^1, the check valve C^1 and the discharge pipe E.

The mathematical analysis of the system is difficult. Any one interested in a full discussion of the problem will find it fully brought out by Prof. Elmo G. Harris in Vol. 54 of *Transactions* of American Society of Civil Engineers.

In order to proportion a plant correctly, the needed data are as follows:

Q_w = the quantity of water to be pumped in cubic feet per second.

h = the total lift, or head, in feet.

l = the total length of air pipes, the distance from compressor to pump tanks.

Having these, we must compute:

Q_a = the volumetric capacity of the compressor, or the piston displacement in cubic feet per second.

V = the volume of each pump tank (cubic feet).

d = the diameter of air pipe (in inches).

D = the diameter of water pipe (in inches).

H. P. = the maximum horse-power required of the steam end of compressor.

From the formula:

$$Q_a = Q_w [1 + 1.4 \log_e R]$$

the following table has been prepared, showing the size of compressor, pipes, etc., required for various heads, based on 100 gal. of water per minute; for other quantities, the dimensions will be directly proportional. The table assumes the pump tanks to be fully submerged.

DIMENSIONS FOR RETURN-AIR SYSTEM.

Lift in feet	Capacity of Compressor in cu. ft. per minute Piston Displacement for 100 gal. per minute	Max. I. H. P. of Air Cylinder	Max. I. H. P. of Steam Cylinder	Average H. P. of Steam Cylinder	Area of Air Pipe in sq. in. for each 100 gal. Capacity of Plant	Area of Water Pipe in sq. in. for each 100 gal. Capacity of Plant
50	39.84	2.74	3.22	2.80	.96	7.70
60	42.78	3.28	3.85	3.37	1.03	8.25
70	45.30	3.85	4.53	3.93	1.09	8.73
80	47.70	4.45	5.22	4.49	1.14	9.12
90	49.80	5.03	5.91	5.05	1.20	9.60
100	51.84	5.67	6.67	5.61	1.25	10.00
110	53.64	6.31	7.43	6.17	1.29	10.30
120	55.44	6.96	8.18	6.73	1.33	10.60
130	57.00	7.62	8.97	7.29	1.37	10.95
140	58.50	8.30	9.75	7.85	1.41	11.30
150	59.94	9.00	10.60	8.41	1.44	11.50
160	61.38	9.75	11.45	8.98	1.47	11.75
170	62.64	10.42	12.25	9.54	1.50	12.00
180	63.84	11.13	13.08	10.10	1.53	12.25
190	64.98	11.85	13.95	10.66	1.55	12.40
200	66.12	12.76	15.00	11.22	1.58	12.65
210	67.20	13.35	15.70	11.78	1.62	12.95
220	68.28	14.09	16.60	12.34	1.64	13.10
230	69.24	14.92	17.35	12.90	1.67	13.35
240	70.20	15.68	18.45	13.46	1.69	13.50
250	71.10	16.46	19.35	14.02	1.71	13.70
260	72.00	17.24	20.25	14.58	1.73	13.82
270	72.84	18.00	21.20	15.14	1.75	14.00
280	73.56	18.80	22.10	15.71	1.77	14.20
290	74.28	19.60	23.10	16.27	1.79	14.30
300	75.06	20.45	24.00	16.83	1.80	14.40

Air-Lift System.

The air-lift system of pumping, measured on the power basis, is not an efficient method of raising water, but where it is and can be applied, it fills a certain field and in that class of work does service that no other pump can do, and does it in a most satisfactory way. This method of pumping has been most generally applied to raising water from artesian wells. There are not many underground formations in which wells can be located close together without affecting each other when pumping, and for that reason it is best to spread them out on a line of what we call the line of underground flow. Some formations are so tight that wells have but little capacity, and in such formations it is particularly necessary that the wells should be scattered and pumped moderately. For the reason that separate pumping units entail high first cost and labor for attendance, the air-lift system is recommended. An economical central pumping station will serve any number of wells, compressed air being capable of transmission in proper sized pipes for great distances without serious loss.

The required equipment consists of an air compressor located at some convenient point, preferably in some existing power station where it can take advantage of economical steam conditions, an air receiver, and a pipe line leading from the receiver to the well or wells. The pump proper consists of nothing more than two plain pipes, one for air and one for water; these pipes run down inside of the well and are submerged for a certain depth proportionate to the lift when the pump is at work. Fig. 3 shows the different methods of piping wells. All wells are not alike, therefore it is a special problem in every case to determine the size and arrangement best suited.

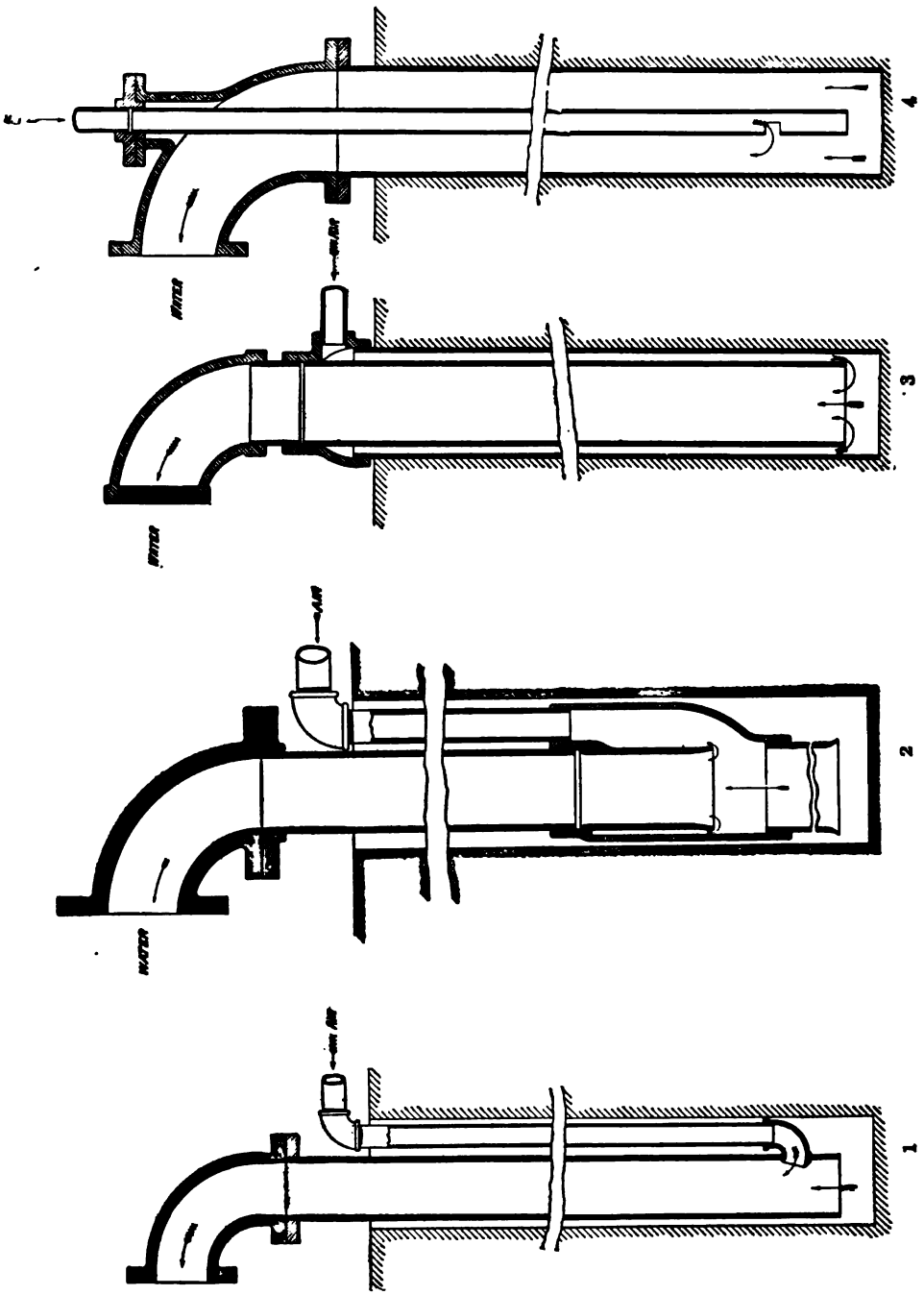

FIG. 3. TYPES OF AIR-LIFT SYSTEMS.

Each method here shown has its use, depending upon the diameter and depth of the well, whether the well is cased or not, the lift, volume of water to be pumped, and character of the water-bearing strata. The arrangement most generally used, where conditions permit, is shown in No. 1. This is the Pohle or "side-inlet" method, in which the discharge and air pipes are placed side by side in the well, joined by a suitable foot-piece. In No. 2, compressed air fills the annular space surrounding the up-take pipe and is free to enter the rising column at all points of its periphery, at the same time acting without obstructing or contracting the discharge pipe anywhere.

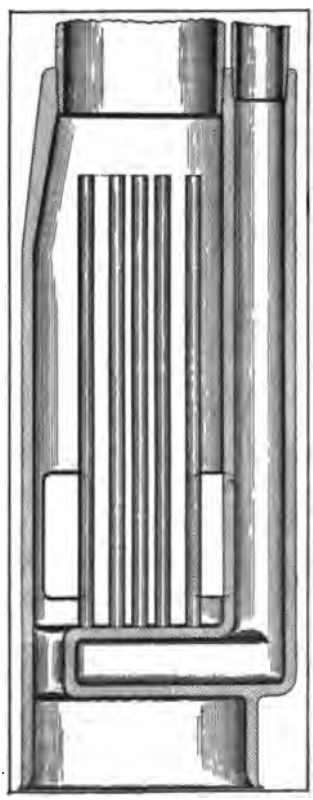

FIG. 4.

Fig. 4 shows the tubular foot-piece. This device aims to provide a means for actually dividing air into fine streams, which we have found to be the best condition in which air can be introduced into the rising main. Air from the supply pipe fills the hollow base of the foot-piece. The base-plate is studded with ¼-in. wrought-iron pipes about 18 in. long extending upwards to a point where the water passes from the foot-piece into the discharge pipe, at which juncture the bubbles of air are released. Ample space is provided for the inflow of water to the foot-piece through the bottom openings, augmented by the larger side inlets shown. Thus we have in the foot-piece designed for a 5-in. discharge pipe a subdivision of air into thirteen ¼-in. streams, with the added advantage that these separate small pipes will tend to act as governors to control the discharge of the air.

The usual capacities allowed for the side-inlet method are as follows:

Air Pipe Connection	Water Pipe	Size Well	Maximum Economical Capacity on Moderate lift. Gal. per min.
½ in.	1 in.	3 in.	7
¾	1½	4	20
1	2	4½	35
1	2½	5	60
1¼	3	6	100
1½	3½	7	140
1½	4	8	190
1½	5	9	250
2	6	10	350

No. 3, Fig. 3 shows the Saunders system. A central discharge pipe is suspended in the well, the air passing down between it and the well casing. If the well is not cased a second pipe must be used outside of the main discharge pipe, the air as before filling the annular space between the two pipes. In estimating pipes under this system, it is usual to allow capacities as follows:

Lift, 25 ft., 15 to 20 gal. per min. per sq. in. area water pipe
Lift, 50 to 125 ft., 12 to 15 " " " " " " " "

No. 4, Fig. 3 is the so-called "central air pipe" system. This method is generally used where the lift is low, and to obtain the greatest possible output from what are termed "strong wells," *i. e.*, wells having a strong static pressure and finding their supply in gravel or open rock formations. The air pipe is suspended in the well without the usual discharge pipe, thereby making the well its own discharge pipe. The proper sized air pipe for different sizes of casing and capacities are about as follows, depending upon the lift and submergence:

Size of Casing	Size of Air Pipe	Capacity. Gal. per min.
3½ in.	1¼ in.	80 to 100
4	1½	100 to 150
5	2	150 to 250
6	2	275 to 375
8	2½	500 to 665
10	2½	775 to 1000

Notwithstanding the fact that the air-lift system has been in commercial use for nearly 20 years and thousands of plants are in daily operation all over the world, no mathematically correct formulas have been published covering the design of these pumps, nor is there exact information at hand for calculating the volume of air and pressure required to operate them under all conditions. This is due to a number of variable factors which enter into each proposition, and if we are to undertake to derive a practical working formula, we must assume certain efficiencies for the system, and assume also that the wells will be properly piped to meet the proper relation of submergence to lift.

Although impossible to construct a formula which can be applied uniformly to all conditions met in this class of work, yet we can define the trend and introduce certain rules which will permit any engineer to calculate the requirements for any given proposition when the exact data covering the case are known. The data required, and the factors entering into these calculations may be reduced to the following:

V_a = free air piston displacement required to raise 1 gal. water.

h = total vertical lift in feet.

H = submergence in feet.

C = constant (efficiency of the system).

Before proceeding we must obtain a clear understanding of what is meant by the symbols above written.

V_a is the free air (piston displacement of the compressor) required to raise 1 gal. of water, due allowance having been made for the ordinary volumetric efficiency of the air compressor. If the compressors are in bad shape or poorly designed, of course more displacement will be required from such a machine than from a well-designed compressor in good working order.

By "lift" we mean the total vertical lift from the pumping level of the water in the well to point of discharge.

"Submergence" is the depth the air pipe is submerged below the pumping level of the water in the well, and is an important factor in any installation; the percentage of submergence, so often heard of, is the percentage of the total length of pipe which is submerged in the solid water when pumping. The necessary percentage of submergence varies in accordance with the lift; low

lifts require proportionately more submergence than high lifts, or in other words, the necessary submergence decreases as the lift increases. The range of these percentages lies within the following limits: For lift of 20 ft., 66 percent.; for lift of 500 ft., 41 percent. The average best percentage in the class of work usually encountered will lie between 50 and 65 percent. Submergence also governs the pressure required, both the starting pressure and the working pressure. The air-lift differs from other pumps in that the pressure necessary to raise the column of water does not depend upon the head, h, but upon the submergence, H, or in other words, H is the actual head against which the air pressure acts.

The first step in a calculation of this kind is to determine from the lift what the proper submergence should be, and knowing this submergence, the work can be easily calculated on the basis, $P = 0.44\ H$.

We are now ready to calculate the air volume required (free-air piston displacement) to raise one gallon of water. The following formula may be used, which closely approximates average practice:[1]

$$V_a = \frac{h}{C \log \frac{H+34}{34}}$$

The annexed table also shows the value of C calculated from the efficiency of the system under various heads, with proper submergence.

TABLE FOR VALUE OF C.

h = Lift	C = Constant
10 ft. to 60 ft.	245
61 ft. to 200 ft.	233
201 ft. to 500 ft.	216
501 ft. to 650 ft.	185
651 ft. to 750 ft.	156

Problem.—Given a bore-hole, well, or shaft 250 ft. deep, the water in which stands 50 ft. below the ground surface, but which falls 25 ft. when being pumped at the rate of 200 gal. per min., and it is required to raise the water 25 ft. above the ground:

Standing water level in well, measured from surface	50 ft.
Drop when pumped	25 ft.
Elevation above surface to raise water	25 ft.
Total lift	100 ft.

[1] In part suggested by Edward A. Rix, A.S.M.E.

If the best ratio of submergence for a lift of 100 ft. is 58 percent., then

$$H = \frac{h \times 0.58}{100 - 58} \text{ or } \frac{100 \times 0.58}{100 - 58} = 138 \text{ ft.},$$

and applying our formula,

$$V_a = \frac{100}{C \log \frac{138 + 34}{34}},$$

and further introducing the value of C from the table, we have

$$V_a = \frac{100.}{233 \times 0.704151} = 0.60.$$

$V_a = 0.60$ cu. ft. (piston displacement) per gallon of water; hence for 200 gal., $200 \times 0.60 = 120$ cu. ft. will be required.

The next calculation is to find the starting and the working pressure. Observe that the normal or standing water level is 25 ft. higher than the working, which indicates a submergence at the beginning of operations equivalent to $138 + 25 = 163$ feet. The starting pressure must be just sufficient to overcome that due to the height of the water column above the foot-piece.

$163 \times 0.44 = 72$ lb. gage, the starting pressure.

$0.44\ H = 138 \times 0.44 = 61$ lb., the working pressure.

The actual pumping level in a well or group of wells can seldom be known in advance of a test. It is customary to assume certain conditions of lift and submergence, basing such conclusions on experience, and proceed to pipe the wells accordingly. After the piping is installed and the working conditions arrived at, the submergence is altered to suit by raising or lowering the pipe in the well until the best ratio is established.

The quantity of water a well will yield depends upon its diameter and the natural flow. It is also essential that the pipes both for air and water be properly proportioned, for the reason that velocity of flow is an important factor; if too large a pipe is used the air gets through without doing all the work it should, and on the other hand, if the velocity is too high, there is undue friction loss, and also inefficient expansion of the air bubbles. The best results are ordinarily obtained by providing for a discharge of 12 to 15 gal. per square inch area of discharge pipe.

Tables are available for determining the friction loss of air pipes. It is, of course, essential to keep this loss within reasonable limits. The flow should not exceed 1,500 ft. per minute.

One reason why the air-lift has been so popular, aside from its flexibility and simplicity, is the fact that the yield of a well is frequently increased by its use. Wells can be cleaned of sand, keeping open the water-bearing seams. As there is little apparatus in the well, an unobstructed flow is assured. The air-lift will handle all the water a well yields, but if a given well is equal, at its maximum, to a flow of 50 gal. per minute, the air-lift has no miraculous power whereby the yield can be increased.

Water is also purified and improved by the intimate commingling with air on its passage through the discharge pipe. There is a complete aeration as the water flows upward, putting it in condition to throw off sulphur gas, precipitate iron, and in a measure prevent vegetable growth.

Special Uses for the Air-Lift.

The pumping of oil wells offers a superior opportunity for the air-lift, and in this line of work compressed air has been preeminently satisfactory. One of the advantages possessed by the air-lift, whether it be in pumping water or oil, is the ability of the system to handle sand and gritty matter.

On account of the facility with which the air-lift, owing to the fact that no working parts come in contact with material being pumped, can handle sand, mud and other substances destructive to ordinary mechanical pumps, it is an ideal method for pumping in the following cases:

1. Slimes, a mixture of water and the very fine portions of crushed ore coming from a reduction plant.

2. Sand and water for stone saws. All preparation necessary for work of this kind is a "master" well or pit, sunk to a proper depth to provide the necessary submergence.

3. The air-lift has done excellent work as a dredge pump and if its usefulness along these lines were better appreciated, we should see a great many more installations. The arrangement of air-lift units on a scow, with proper facilities for raising and lowering the pipes, presents unexcelled features of simplicity and capacity.

4. The agitation of slimes in the cyanide process also offers an opportunity for the air-lift, as complete and satisfactory aeration can be brought about by placing an air-lift in a cyanide tank specially designed for the purpose.

The air-lift is past the experimental stage, and is taking its place with other applications of compressed air where steady, reliable results are appreciated.

BOOK REVIEWS.

Stamp Milling. A treatise on practical stamp milling and stamp mill construction. By Algernon Del Mar. New York, McGraw-Hill Book Company, 1912, x + 134 pages. Cloth, $2.00 net.

This work, to a great extent, describes the practical working experience of Mr. Del Mar himself and the construction work with which he has been associated; he states in his introduction that he hopes that this collection of experiences will stimulate the student to studies along the same line of work.

After briefly considering the evolution and history of the gravity stamp mill and giving the general principles of the stamp mill itself, the writer, in Chapter II, makes an interesting comparison between the work of heavy stamps as installed at the present time and the lighter stamps of the California type. He draws the conclusion that for coarse crushing, to be followed by fine grinding, heavy stamps are preferred but for fine grinding in the stamp mill the lighter stamps give the best results.

The last half of Chapter II and Chapter III are devoted to a detail description of the stamp mill proper and the practical working of the same. As this is the result of actual practice the information given is extremely valuable.

In Chapter IV the limitations of the single unit and of the five-stamp unit batteries are considered. Under this head it is unfortunate that the writer does not give us more information on the subject of the single unit, as he treated it from the standpoint of amalgamation rather than from the standpoint of a crushing machine.

Chapter V gives a short history of amalgamation, and Chapter VI is devoted to amalgamation itself. In the last chapter some very interesting information in regard to amalgamation practice is noted.

Stamp mill construction is fully discussed in the next and last chapter, and from both the text and illustrations the reader can obtain valuable information in regard to the construction of our modern stamp mills. The illustrations and description of South African installations are excellent.

The book is well printed, and the complete index will be found very useful. A. L. W.

Addresses to Engineering Students. With an introduction by J. A. L. Waddell and John Lyle Harrington, Consulting Engineers. Kansas City, Mo., Waddell & Harrington, 1911. ix + 493. Cloth, 75c. (when shipped in large numbers).

This work, as the name indicates, is a collection of a number of addresses made by prominent engineers to engineering students. In reading these addresses, one cannot help but feel that even if only some of them were read by men who are taking the engineering courses at our various institutions, the benefit to the reader would be very great and he would be able to understand more about the life he is

about to enter. He would learn what is meant by engineering, the advantages of the engineer's profession and the responsibilities an engineer assumes. He would also see that above all an engineer should be a gentleman.

There are so many good things that might be quoted from this book that it is impossible to make a selection. But among the addresses it will not be amiss to call special attention to the one made by the late Walter C. Kerr, entitled "The Point of View," and to the admirable advice which he gives in this address. A. L. W.

The Theory and Practice of Technical Writing. By Samuel Chandler Earle, Professor of English in the engineering school of Tufts College. New York, The Macmillan Company, 1911. 301 pp., illus. $1.25.

A well qualified speaker not long since made the remark that, of all men, the engineer nowadays has the most interesting things to tell, but that he is the very one least able to tell them. A careful attention to the precepts of this volume would be helpful in removing that reproach. The method of treatment is not that of an elementary grammar; the author presupposes a working facility with the written language, and proceeds to develop the peculiar requirements of technical writing, which should be characterized above all other writing by accuracy, completeness, logicalness, and economy of presentation. The author also draws the distinction between those writings intended for the general reader and those for one's technical colleagues, showing wherein the nature of the treatments should differ.

Referring to the writers of so-called "popular science," and their tendency to distort truth for the purpose of attracting attention, the author properly remarks: "To think that science, in order to be popularized, must or may be inaccurate is a serious confusion of ideas. The main purpose of scientific writing is always to instruct. In writing for the uninformed, additional care, if possible, should be taken, for they cannot, like the specialist, correct untruths from their own knowledge."

As a means of visualizing the points under discussion, a variety of articles taken from scientific and technical publications are printed in an appendix, to which the reader's attention is directed by copious cross-references.

The author's own style shows a few defects, the more noticeable, perhaps, on account of the subject of the book. One of them is the recurring use of the awkward circumlocution "there are" as the opening words of a sentence; this is an exceedingly common construction, but is the more inexcusable because usually unnecessary. Another defect is the repeated use of the unsupported adjectives "some" and "most," meaning "some writers," or "a majority of readers."

Of the several books on this subject that have come under our notice, this is the most practical and helpful, and will repay careful study even by an experienced writer. E. K. J.

The Autobiography of John Fritz. New York, John Wiley & Sons, 1912. xvi + 326 pp. Cloth, $2.00 net.

As the author states, this book is dedicated to the loyal, able and fearless men who have faithfully stood by him throughout his career, and in this dedication, in simple language, he gives them credit for the work they have done in rendering this assistance.

It is impossible to read the story of the work done by a man like Jonh Fritz without being greatly impressed, and all who read this work and follow the various events in his life from the beginning to the present time, cannot but realize what an important part John Fritz and his work have played in the metallurgy of iron and steel, and what great assistance he has rendered in advancing that art to its present state of perfection.

Commencing in the first chapter with his boyhood life and early days at school, we realize that evidently he knew that the keynote of success is hard and assiduous work, for at all periods in his early career he used this motto as his creed, and spent all of his spare time in studying conditions as they were.

The chapters on his experience at Cambria, in which the installation of the first 3-high roll train is described, and also the immediate destruction of this plant by fire, after it had been put in commission, places before us a thrilling incident, and one which would have unnerved many men. But without loss of time the 3-high mill was placed in operation again and a building erected around it.

The following chapters deal with his work at Bethlehem, and then a most interesting description of the bessemer and open hearth processes, the blooming mill, structural-steel plant and armor-plate mill is given.

On reading this book one cannot help but feel that he has been brought into close association with the life and works of a great man, and that in becoming acquainted with his work he is given a clearer insight into the metallurgy of iron and steel, and what has been done in this connection in the past seventy years.

The last portion of the book is devoted to a description of the various honors conferred upon John Fritz, the different dinners which have been given to him and numerous testimonials bestowed on him in appreciation of the great work he has done.

A. L. W.

PERSONAL ITEMS.

E. L. Dufourcq, E. M., '92, consulting engineer, of New York, was recently engaged in professional work at the Sovereign mine, Ladies Canon, Sierra county, Cal.

M. B. Spaulding, E. M., '95, has recovered from his serious accident of a year ago, and is now engineer in active charge of the Calaveritas Consolidated placer mines, near San Andreas, Calaveras county, Cal.

John D. Irving, Ph. D., '99, is senior author of Bulletin 478, of the U. S. Geological Survey, recently issued, treating of the geology and ore deposits near Lake City, Colo.

Prof. Arthur L. Walker, E. M., '83, contributed a review of the progress in copper smelting during 1911 in the *Engineering and Mining Journal,* Jan. 6, 1912.

Howard R. Stewart, E. M., '02, was married on Jan. 2 to Lesa Beryl Ball, of Putney Heath, London, England.

At the last annual meeting of the Mining and Metallurgical Society of America, J. Parke Channing, '83, was reëlected President; and J. F. Kemp, '84, H. S. Munroe, '69, and T. H. Leggett, '79, were elected to the council.

Heinrich Ries, Ph. D., '92, is senior author of Memoir 16-E, recently issued by the Canada Department of Mines, Ottawa, on the subject of clay and shale deposits of Nova Scotia and parts of New Brunswick.

S. F. Shaw, E. M., '03, publishes in the *Engineering and Mining Journal,* for Jan. 13, 1912, a very valuable compilation of data covering tonnage, profits, dividends, costs, and reserves of 275 of the more prominent mines of the world.

W. H. Hendrickson, E. M., '03, has been appointed manager of the Horn Silver mine, at Frisco, Utah.

L. D. Huntoon, E. M., '95, contributed a review of progress in the stamp milling and gold ores during 1911 in the *Engineering and Mining Journal,* Jan. 6, 1912.

H. B. Barling, E. M., 1903, has been appointed manager of the New York and Honduras Rosario Mining Company, at San Juancito, Honduras.

A. P. Rogers, E. M., 1898, has recently been engaged in professional work in Arizona.

Alonzo F. Bardwell, E. M., 1883, has bought the entire business of Bettles & Bardwell, assayers and chemists, of Salt Lake City, since the death of his partner.

Don Gelasio Caetani, E. M., 1903, has entered partnership under the firm name of Burch, Caetani & Hershey, consulting mining and

metallurgical engineers, with offices in the Crocker Building, San Francisco, Cal.

Recent alumni visitors at the School of Mines have included: E. D. Self, '94, late of London and South Africa; J. I. Kane, '10, of El Paso; J. E. Rypinski, '11, of Bingham Canon; W. H. Sherman, '10, until recently with the Foundation Company of America; H. W. Aplington, '03, with the bridge department of the New York Central; S. F. Shaw, '03, of New York; and B. B. Gottsberger, '95, General Manager of the Miami Copper Company, Miami, Ariz.

The many friends of Professor Kemp will be interested to hear of the honors that have been coming his way. At the annual meeting of the American Institute of Mining Engineers, in February, he was elected President for the ensuing year; for a number of years past he has been serving on the board of directors. In March, he attended, as a delegate, the meeting of the Canadian Mining Institute, at Toronto. Professor Kemp was appointed by President Butler to represent Columbia University at the centennial exercises of the Philadelphia Academy of Science later in March; on this same occasion he likewise acted as delegate from the Sociedad Scientifica Antonio Alzate, of Mexico, of which he has been corresponding member for some years.

Percy M. Burrill, E. M., '96, has returned from Europe, and is now located in New York.

George B. Farish is superintendent of the Dobie gold mine, at Porcupine, Ont.

John T. Roberts, Jr., E. M., '07, is superintendent of the Wanakah Mining Co., at Ouray, Colo.

E. H. Messiter, C. E., '94, has patented an electric weighing device which is proving to be a commercial success, and is being adopted quite widely. Office is at 180 Thirteenth Ave., New York.

A. A. Leach, E. M., '11, is on the geological staff of the Cananea Copper Co., Cananea, Sonora, Mexico.

T. M. Bains, Jr., E. M., '11, is in charge of the operations of the American Zinc Co., at Mascot, Tenn.

R. VanA. Mills, E. M., '06, lately passed through New York on his return from China. He has been serving as Professor of Mining at the University of Tsin-Tsin, which is now closed indefinitely by the revolution. Mr. Mills may become attached to the Bureau of Mines, at Washington, D. C.

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THE SCHOOL OF MINES QUARTERLY.

A JOURNAL OF APPLIED SCIENCE.

CONTENTS

	PAGE
ELECTRIC MINING APPLICATIONS. By *E. A. Lof*	317
PHYSIOLOGICAL EFFECTS OF CARBON MONOXIDE. By *Henry S. Munroe*	340
STATIC AND DYNAMIC FORCE RELATIONS IN MACHINES. By *W. Rautenstrauch*	345
CAISSON FOUNDATIONS FOR THE SEAMEN'S CHURCH INSTITUTE BUILDING. By *Maurice Deutsch*	358
PARKS ELECTRO-CYANIDE PROCESS. By *John R. Parks*	365
RARE MINERALS IN CALIFORNIA. By *Austin F. Rogers*	373
LIMONITE DEPOSITS OF STATEN ISLAND. By *C. R. Fettke*	382
ABSTRACTS—ANALYTICAL CHEMISTRY. By *E. Waller*	392
ABSTRACTS—METALLOGRAPHY AND ALLOYS. By *W. Campbell*	402
BOOK REVIEWS	409
PERSONAL ITEMS	416

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THE
SCHOOL OF MINES
QUARTERLY

VOL. XXXIII. JULY, 1912. No. 4.

ELECTRIC MINING APPLICATIONS.

BY E. A. LOF.*

The importance of the coal mining industry of United States can best be realized by reference to the statistics, wherein it appears that the coal production in 1880 amounted to less than 70 million tons, but in 1910 had increased to over 500 million tons per year. The value of this output increased at an even higher rate, due to increased operating expenses. As the mines grew larger and deeper, expenses increased until, in some cases, they could not be operated profitably any longer. This led ultimately to the introduction of electricity for mining operations, and it is now generally conceded that this system is much superior to any other. Almost all new mines are being equipped with electric drive, and a very large number of old mines are changing over to this system. Not only does this reduce the cost of working, but it also offers a much safer and reliable operation.

ADVANTAGES OF THE ELECTRIC SYSTEM.

A single generating station can be used for feeding a whole mining district, resulting in a much improved load-factor and a corresponding reduction in the cost of power. The station can be centrally or most economically located with regard to water and fuel supply. Large, modern, steam-turbo generator units can be in-

* Power and Mining Engineering Department, General Electric Co., Schenectady, N. Y.

stalled, resulting in a most efficient generation and continuity of service.

Power may also be purchased from existing transmission systems, or available water powers may be developed and the energy transmittd to the mines. This latter method is now actually being carried out at some large coal fields, where the water supply at the mines is undesirable for use in boilers. If the distance over which hydraulic power has to be transmitted to the mines is not too great, a saving may also be accomplished by the use of water power, especially if a good market price is obtained for the coal, which would otherwise be consumed at the mine.

The economy with which electric power can be distributed to the various points in a mine surpasses all other methods. The electric system eliminates long and expensive steam and air lines, with which the danger of breakdown and the difficulty of keeping up the necessary working pressure increases for every extension to the service. Electric distribution, on the other hand, is most simple and flexible. Very large districts can be efficiently supplied, and additions or alterations can at all times be made without the least difficulty.

A most efficient application of motors to the different mining machines is readily accomplished. They can be direct connected, or geared to the driving shafts, thus reducing the friction losses and repair charges to a considerable extent, while, on the other hand, the cost of belting and countershafts is entirely eliminated. Individual motors can be substituted for driving conveyors, scrapers and other machinery in breakers and tipples, which formerly were equipped for group operation by means of inefficient engines. In motor-driven breakers, the saving in belting alone is considerable.

Operation with the electric system is very simple, and results in a materially increased output of a mine. Perfect control is at all times possible. Simple, automatic, safety devices can be installed, and indicating or recording meters can be provided in the different circuits as desired, and the performance of every individual machine ascertained. This is a very important point, as it is possible to maintain the machinery in the best operating condition. Any excess consumption of power can at once be detected and the defect remedied, while also an accurate record can be kept of the cost of the different operations.

Both alternating and direct current are used for mining work, generally depending upon the particular conditions of the project. For smaller installations, with short distributing lines, the direct-current system can be used, the direct-current motor being well adapted for the operation of hoists, mining locomotives, blowers, etc. When, on the other hand, power has to be distributed over greater distances, as in larger mining districts with scattered shafts, the alternating-current system will generally be found more advantageous. Due to the possibility of using a higher voltage, a more economical distribution can be obtained. The simplicity of the squirrel-cage induction motor makes it especially suitable for the severe service met in mining work. The absence of slip rings and sliding contacts eliminates sparking, with the accompanying danger of firedamp ignition. For the operation of mining locomotives or other machinery requiring it, direct current can readily be obtained by motor generators or rotary converters, so that in general it can be said that the alternating-current system offers many advantages for mining work.

The advantages of electric lighting are too well known to need comment. In many mines in which the electric system has not been utilized for power, it has, however, been used for general illumination, and with the introduction of reliable and efficient miners' lamps the electric system should be still further appreciated.

HOISTING.

The electric motor affords an ideal method of applying power to hoists, due to its rotary motion and high torque characteristics. Its use for this purpose has numerous and well defined advantages as compared with steam and air systems. Complete control is afforded over the hoist when running, enabling the operator to work much quicker and with greater certainty. The ease with which brakes can be operated electrically makes it possible to manipulate the hoist with the greatest precision, the mechanical brakes being used only for holding the load. In certain circumstances, energy may be returned to the system by employing a system of regenerative braking, so designed that the weight of the descending skip may be utilized to drive the motor as a generator, and thereby feed back an appreciable amount of current into the distribution system.

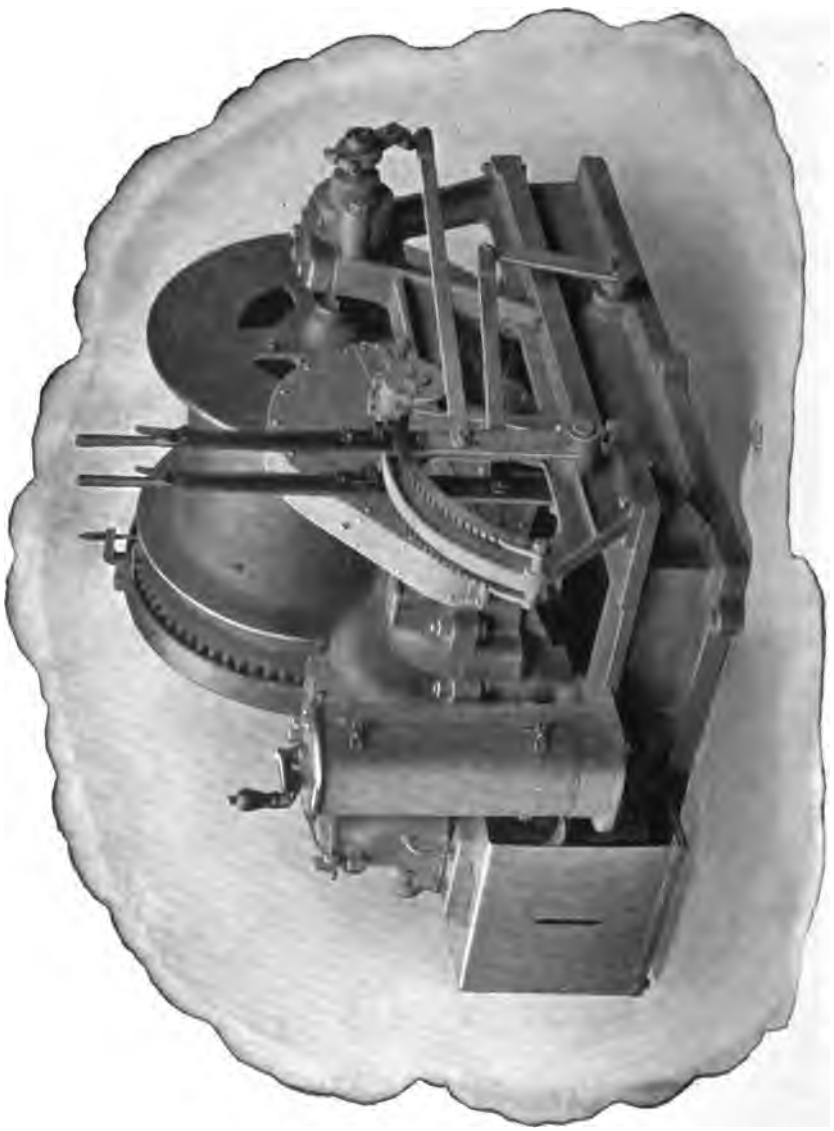

FIG. 1. MOTOR-DRIVEN MINE HOIST.

Power consumption is thus considerably reduced, besides obviating excessive wear on the brakes which would otherwise have to absorb the energy.

The application of steam engines to hoisting has many serious disadvantages, among others that due to the intermittent service generally required, which necessitates certain precautions in resuming the operation of a steam hoist after it has been at rest. If water collects in the cylinder, it must be thoroughly drained before starting to hoist, to avoid the danger of blowing out a cylinder head. In cold weather the operation is frequently complicated by excessive condensation, or the formation of ice. In the electric system, these difficulties are of course entirely eliminated.

Objections to air-operated hoists are numerous, and compared to them the electric system offers many important advantages besides a lower cost of operation. The number of hoists that can be served from a central compressor plant is limited, owing to the expense of piping. Extensions of an electric system can be made with less difficulty and at less expense and protective devices, insuring a safe and reliable operation, are readily installed. Complicated valve mechanisms, cooling water for compressors, and reheaters for the engines, make the air system objectionable, while likewise its efficiency decreases in a very short time, due, to leaky valves, pistons and air mains, unless extreme care is taken to guard against these losses, which in turn necessarily increases the charges for repair and maintenance. The characteristics of the electric hoist motor are such that its speed is automatically limited to a predetermined value, without the use of auxiliary devices, thus reducing to a minimum the possibility of a runaway when lowering unbalanced.

It has been estimated that the annual cost of operating an electric hoist is much less than that of an equivalent compressed-air hoist, and that the saving realized by the use of electric hoists will pay for the complete installation in from five to six years. For isolated hoists, the maximum demand of power is greater for the compressed-air system. This is a very important item where power is purchased and the peak-load is penalized; here the economy in the use of an electric hoist with equalizing fly-wheel sets will be still further marked. Where compressed-air systems have been installed this

fact has nevertheless generally been recognized, and the reasons for choosing the air system have been mostly a matter of first cost, in that some of the old machinery could be retained, which would not have been the case had the electric-hoist system been selected.

For driving electric hoists, shunt-wound, direct-current motors are very largely used, owing to the wide range of speed control which can be obtained economically with this type of motor. In many of the later installations, however, the polyphase induction

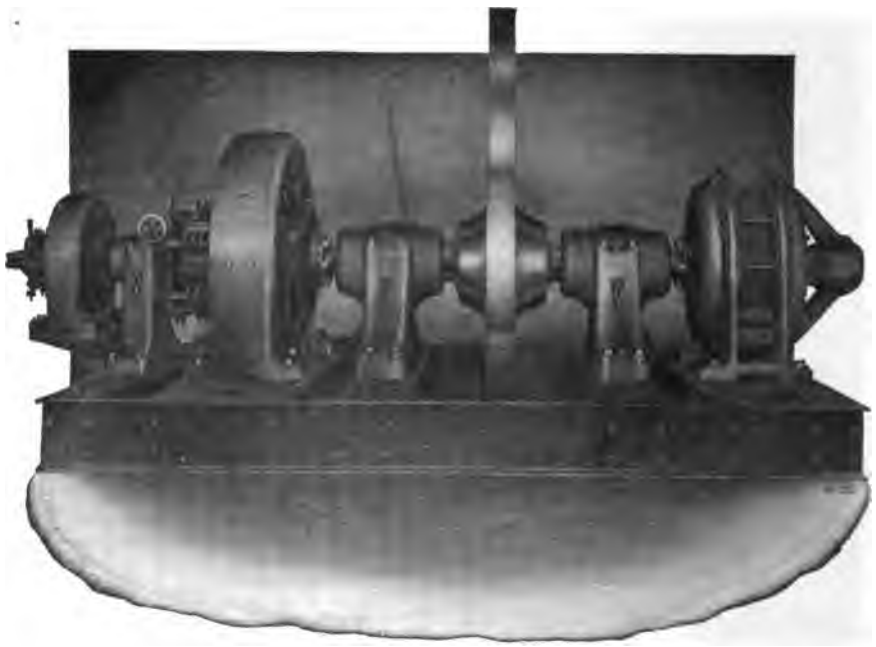

FIG. 2. FLYWHEEL MOTOR-GENERATOR EQUALIZING SET FOR MINE HOISTING.

motor with a resistance rotor control has been applied to this service with entire success; simple and thoroughly reliable controllers can readily be provided to secure the variations in speed required for mine hoisting.

Where the source of supply is alternating current, as is almost invariably the case, and it is desired to use a direct-current hoist motor, a motor-generator flywheel set is generally employed as a medium. This system is generally known as the Ilgner system and employs the Ward-Leonard control. The connections for this system are shown in Fig. 3, from which it is seen that the hoist is

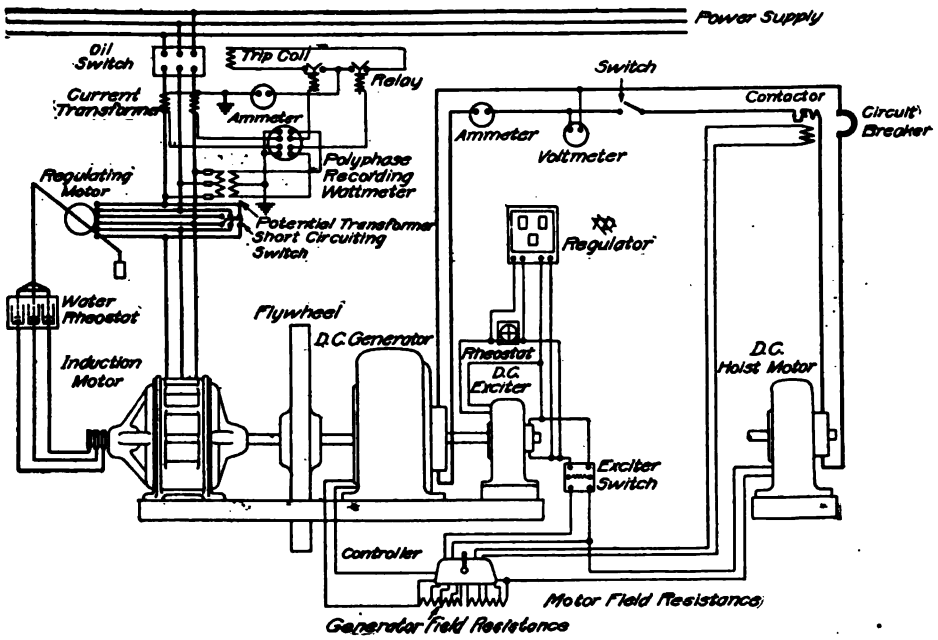

FIG. 3. CONNECTIONS OF FLYWHEEL MOTOR-GENERATOR HOISTING EQUIPMENT WITH REGULATING MOTOR IN MAIN LINE.

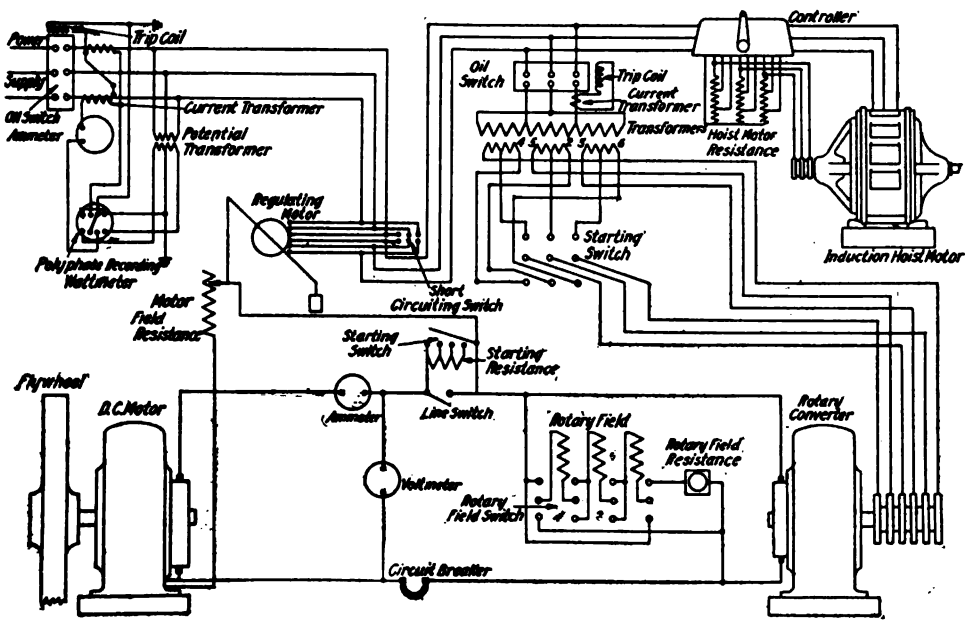

FIG. 4. CONNECTIONS OF INDUCTION HOIST MOTOR WITH FLYWHEEL ROTARY CONVERTER EQUALIZER EQUIPMENT.

driven by a shunt-wound, direct-current motor, receiving power from the alternating-current supply system through an induction motor-generator set to which is coupled a heavy flywheel. The speed of the hoist motor is controlled by varying the voltage of the direct-current generator, which is separately excited from an exciter mounted on the shaft of the set. One generator is therefore necessary for each hoist motor, although one induction motor may drive two generators. In order to permit the flywheel to take care of the peaks, and equalize the load, the speed of the set must be varied according to the demand of power and this is accomplished by an automatic regulator connected in the secondary circuit of the induction motor. In its most common form this regulator consists of a water rheostat. The resistance is varied by means of movable electrodes suspended from an arm mounted on the shaft of a small induction motor which is set on the top of the rheostat tank. The regulator motor is connected in series, either directly or through series transformers, with the induction motor of the flywheel set. It is adjusted so that its torque opposes the weight of the electrodes, which are partly counterbalanced to reduce the size of the regulator motor to a minimum, and to permit adjustment of the regulator for different values of the line current. When this exceeds the value for which the regulator is adjusted, the torque of the motor overbalances the weight of the electrodes, lifting them, thus inserting resistance in the rotor circuit of the induction motor of the set. This causes it to slow down, and allows the flywheel to assist in driving the generator during the peak load.

Where it is desired to drive the hoist by a polyphase induction motor, the simplest system is the one in which the motor is simply direct-connected or geared to the hoist drum, without the use of any equalizing set. The motor is always of the phase-wound type and the speed is controlled by a rheostat which is connected by means of the collector rings to the secondary or rotor circuit. It is generally of the liquid type on account of the heavy currents involved, and consists of a sheet-iron tank divided into two compartments; the upper contains the electrodes and the lower contains coils for cooling the salt water which is kept circulating continuously from the cooling tank to the rheostat compartment by a centrifugal pump. The resistance is varied by changing the level

of the liquid in the upper tank, and this in turn is done by operating a weir or gatevalve, which permits water to flow out and lower the level if the resistance is to be increased and vice versa if it is to be decreased. The gate is manipulated by levers conveniently located for the operator.

The power taken by this system is constant during the period of acceleration, but the efficiency for this period is very low. No

FIG. 5. AUTOMATIC LIQUID RHEOSTAT FOR INDUCTION MOTOR SPEED VARIATION.

power is returned to the supply system during retardations, but only when lowering the empty skip, and the power consumption for small movements of the cage or skip is very large. On the other hand, the efficiency during the period when the hoist is running at full speed is high, and no power is consumed while the hoist is at rest. The efficiency of the complete cycle of operation decreases rapidly with a decrease in the time during which the hoist is driven

at full speed, while it increases with an increased rate of acceleration.

With the Ward-Leonard system first described, the power consumed during acceleration is much less than with the induction-motor hoist and a considerable part of the energy stored in the revolving parts of the hoist is returned to the system during periods of retardation, and also when lowering the empty skips. When the hoist is running at full speed, however, the efficiency is lower, and it must also be taken into consideration that the losses in the motor-generator set must be supplied during the time when the hoist is at rest. It follows, therefore, that the Ward-Leonard system is more advantageous for short lifts, in which case the period of acceleration is a large percentage of the total hoisting cycle, and the time during which the hoist is idle is a minimum.

The induction-motor hoist system, previously described, is however objectionable on account of the fluctuation in the power demand, to overcome which a converter equalizer is sometimes added. Such a combination is shown in Fig 4, and includes a rotary converter connected on the alternating-current side to the supply system, and on the direct-current side to a motor driving a large flywheel. A regulator actuated by the line current controls the direct-current motor field, so that when the power taken by the hoist drops below the average, the field is automatically reduced, causing the flywheel set to speed up and absorb power from the supply system and store it in the flywheel. When the load on the hoist motor exceeds the average, the operation is reversed, the flywheel set slows down and power is returned to the system through the rotary converter.

Either of the two flywheel equalizing systems here described can be used where the supply is direct current; in the first system by substituting a direct-current motor for the induction motor, and in the second by omitting the rotary converter, but retaining the motor flywheel set.

HAULAGE.

The most efficient and satisfactory method of haulage is by electric mine locomotives. Actual experience has shown that the cost of hauling is from 50 to 75 percent. cheaper by electric locomotive

than by mules. A consideration of the number of mules and drivers required to perform the same work as a locomotive makes this evident.

The direct-current, two-motor locomotive is now generally recognized as the standard type for mine work. There are two general forms of this type; in one the side frames are placed outside of the wheels and in the other the side frames are placed inside of the wheels. For a given track gauge, the outside frame allows maximum space between the wheels for the motors and other parts of the equipment, renders the journal boxes more accessible, and gives somewhat more space for the motorman at the operating end. The inside frame restricts the space between the wheels available for motors and other equipment, but allows for the minimum overall width — a construction very desirable in those mines where props are set close to the track, or the space outside the rails is otherwise limited. The wheels being outside the frame, in case of derailment this type is somewhat more readily replaced.

For hauling cars between the working face of rooms and the entries, gathering locomotives are now recognized as the most

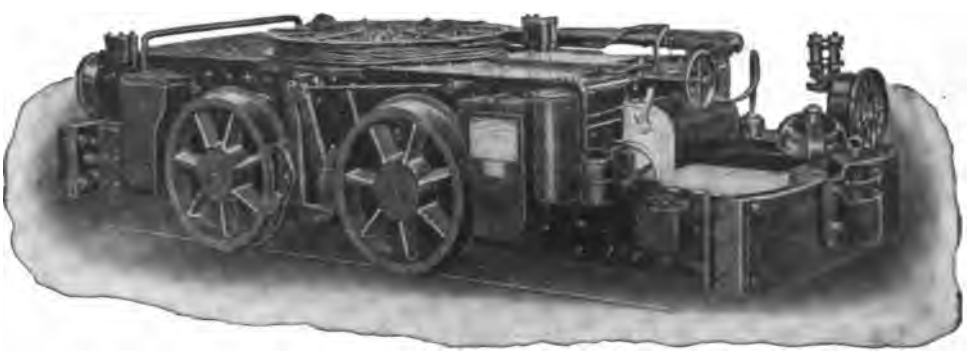

FIG. 6. ELECTRIC GATHERING MINE LOCOMOTIVE.

efficient. In general construction, gathering locomotives are similar to those for main haulage, except that they are usually lighter and are provided with cable reels. The reel is driven through a double reduction gearing by a small, vertical, series-wound motor, the reel being supported by the motor frame and turning on a ball-bearing between the main gear and the top of the motor. The reel is equipped with about 500 ft. of flexible, heavily insulated cable.

The inner end of this cable is connected to a collector ring on the under side of the reel, and the outer end is fitted with a copper hook for attachment to the trolley wire.

On leaving an entry, the cable is hooked over the trolley wire, and as the locomotive moves forward the reel motor is overhauled and acts as a series generator, its counter-torque being sufficient to produce a tension on the cable which causes it to pay out evenly, and drop along the roadbed without kinks. Owing to the braking effect of this counter-torque, the reel ceases to rotate the instant the locomotive comes to a standstill. As soon as the locomotive starts back and slackens on the cable, the motor action comes into play and the reel winds up the cable. The operation

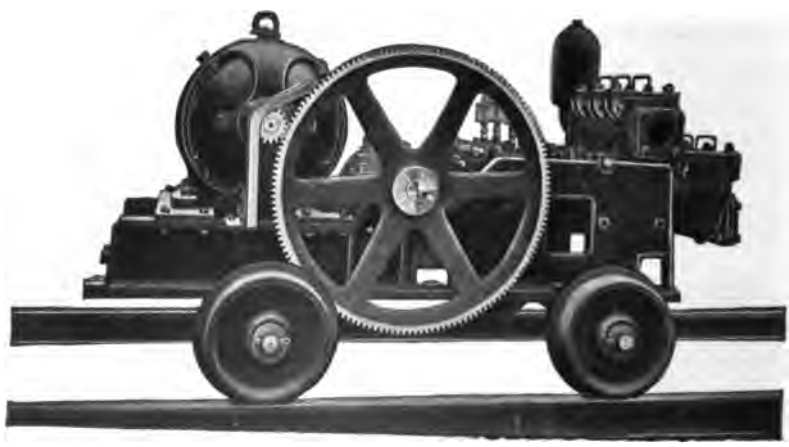

FIG. 7. MOTOR-DRIVEN PORTABLE MINE PUMP.

of the reel is entirely automatic; there are no switches or shifting levers for the motorman to handle, and he is therefore free to devote his entire attention to running the locomotive.

For gangways in which the grade is so steep as to prohibit the use of the cable-reel locomotive, the combination type is preferably used. This locomotive is of the same general construction as the reel type, with the addition of a hoisting drum and steel cable, by means of which the loaded cars are pulled up the slopes and then delivered to the main tracks in the regular way. As this type of locomotive can also perform the duties of the direct haulage and cable-reel types, it is often considered indispensable in mines where a limited number of locomotives are to handle the entire output.

Mining locomotives can also, if desired, be equipped with alter-

nating-current motors. They are of the same general construction as the direct-current locomotives and are equipped with two three-phase induction motors. Their operation requires two overhead trolley wires, with the track rails constituting the third leg of the three-phase circuit. To collect current, two separate trolleys of the standard mine type are used, these being mounted on opposite

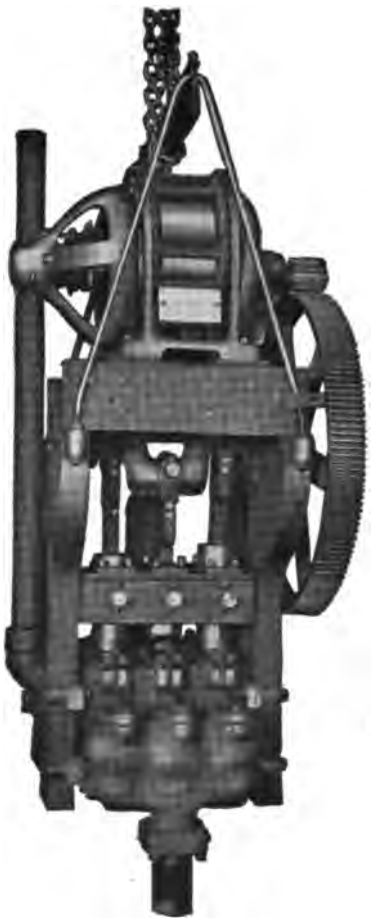

FIG. 8. VERTICAL MOTOR-DRIVEN SINKING PUMP.

sides; but for certain conditions a double trolley, two poles on a common base, can be furnished. As the induction motor is inherently high-speed, a double gear reduction is used.

PUMPING.

For mine work, two classes of pumps are generally used, main pumps and auxiliary pumps. The former are generally of the

stationary type and are used for pumping water out from sumps into which it collects, either by gravity or through the auxiliary pumps. Main pumps are of either reciprocating or centrifugal type, driven either by direct-current or by induction motors. It has been the custom to use centrifugal pumps for low heads and large amounts of water, but recent practice has indicated that these pumps will operate very efficiently and satisfactorily for heads of 1500 ft. and over. The ability with which they can handle muddy and sandy water makes them preferable for mining service, and the smaller space they occupy is well worth considering. They are readily driven by direct-connected, polyphase induction motors, thus eliminating the friction losses of a gear drive.

Auxiliary pumps for raising water from scattered points in a mine on a lower level than the main sump, are mostly of the portable type, although they may be stationary. They are generally driven by direct-current motors, fed from existing trolley circuits. Among such auxiliary pumps, the portable set is of interest. The pump and motor are mounted on a truck of the same gauge as the mine tracks, so that it can readily be hauled by a locomotive to any place in the mine and immediately put into service.

The sinking, or dip pump, may also be mentioned. This type is generally used for pumping out flooded mines. It is designed to be lowered vertically in a shaft and is either mounted on a float or is suspended from the hoisting cables, so as to float on the surface of the water. Squirrel-cage induction motors should preferably be used for these pumps, as they are liable to be entirely submerged, which however will not cause any harm as there are no collector rings nor moving contacts.

Air Compressors.

The air compressors found in mining service are, as a rule, utilized for providing compressed air for air drills. Large air compressors are generally located at some central point in the mine, from which piping is run to the various places where compressed air is required. They are generally of the reciprocating type, due to the high pressure required for drilling, about 80 lb. per sq. in. As they require a comparatively low speed, and are characterized by steady load and low starting torque, the syn-

chronous motor is especially well adapated for driving them. The possibilities that these motors offer for improving the power factor are also of importance in their selection.

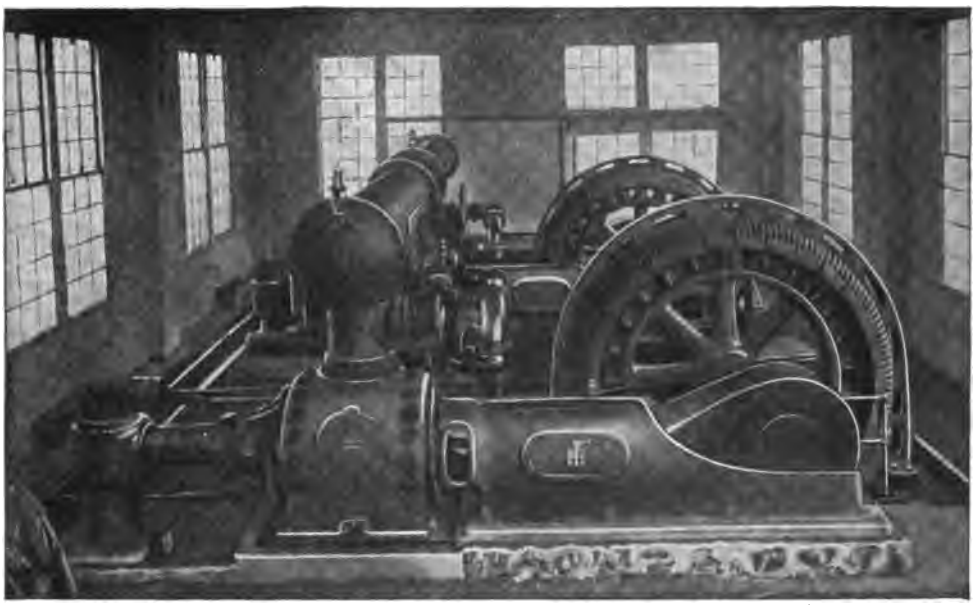

FIG. 9. SYNCHRONOUS MOTOR-DRIVEN AIR COMPRESSORS.

DRILLING.

The problem of making a satisfactory electric rock-drill has been studied for many years, and many more or less successful devices have been invented. The latest development along this line is the motor-operated, rotary-hammer drill. The mechanism of the drill consists of two parts, a revolving helve containing the hammers, and the chuck mechanism for holding and rotating the drill steel. A flexible belt connection between the motor and the drill permits a variation of speed to any desired degree, so that all the advantages of hand drilling can be obtained without the disadvantages usually incident to machine drilling. The drill mechanism is totally enclosed within a heavy cast-steel casing which protects the working parts from any foreign substance, and is of sufficient strength to withstand the severe strains to which a drill is subjected in mining service.

The striking mechanism consists of two steel hammers guided

by a revolving helve. As the helve revolves the hammer is thrown outward by centrifugal force, and at each revolution strikes a blow upon the projecting head of the drill-steel cap which delivers the energy of the blow to the drill. After delivering the blow, the hammer rebounds into the chamber within the helve, where it is completely cushioned upon air which it traps. During the period of recoil the hammer passes the projecting drill-steel cap, as the helve is continuously revolving. The hammer is again thrown into striking position by centrifugal force, during the remaining portion of the revolution.

The drill steel is kept rotating by means of a heavy worm-gear reduction, driven from the helve shaft. A rugged slip-friction cone is mounted on the worm-gear shaft and serves to protect the gears from undue strain in case of a sudden sticking of the drill. When not striking rock, the blow from the hammer is absorbed by buffer plates, which also retain the steel in the chuck while backing out of deep holes in broken and uneven ground.

The motor is fully enclosed and splash-proof, capable of successful operation in wet places, and is reinforced throughout with particular reference to the requirements of rock drill service. The drills can be equipped with either direct-current or three-phase, alternating-current motors. The power required for its operation is about $1\frac{1}{2}$ to 2 horsepower.

The above electric drill has been on the market for only a comparatively short time, and is not as yet extensively used. The "electric-air" drill, however, has been in general use for a number of years and has proved very satisfactory. It is really an air drill driven by pulsations of compressed air created by a duplex air pulsator actuated by a standard electric motor. The air is never exhausted, but is used over and over again, playing back and forth in a closed circuit.

The drill consists simply of a cylinder containing a moving piston and a rotation device, with no valves, chests, buffers, springs, side rods or pawls. The cylinder is larger, but the stroke is shorter, making the weight of the drill unit about the same as that of a corresponding air drill. The pulsator is a vertical, duplex, single-acting, air compressor with opposite cranks, but with no intake or discharge valves nor water jackets. It is geared to a motor, either

direct or alternating current, and mounted on a wheeled truck for easy handling. Two short lengths of hose connect the pulsator with the drill, each running from one pulsator cylinder to one end of the drill cylinder.

The ordinary air or steam driven rock-drill takes a full cylinder of air or steam at full pressure at each stroke, and exhausts it at practically full pressure. No advantage, therefore, is taken of the expansive properties of the air or steam. The electric-air drill, however, operates on an entirely different principle. The closed system comprising the outfit is filled with air under a low pressure, which is simply a transmitting agent between the piston of the pulsator and the piston of the drill itself. The object of slightly compressing this air is to give it a greater density for the transmission of the pulsations imparted to it by the pulsator. In fact, the air in the system may be considered as a cushion between the pulsator and the drill, the pressure in the air simply giving the requisite tension. Practically the only loss of power is that consumed in overcoming the friction of the mechanism, as hardly any loss takes place between the pulsator and the drill.

Coal Cutters.

The cutters used in coal mining are generally of three types, the disc, the bar, and the chain type. Of these, however, the chain type seems to be most extensively employed in this country. The chain is driven by an electric motor and the whole mechanism is mounted on a truck; the whole machine is automatically moved sidewise from one side of a room to the other, thus completing the cutting in one operation.

The motor is especially designed for this work, and is of the multipolar, vertical, shunt-wound, direct-current type, equipped with a ventilating fan. The armature is built on a spider, making shaft replacements simple and economical. The controller and the interlocking reverse switch are of ample size, mounted on a slate base. The machine is so arranged that electric power can be used for unloading, moving to position at face, loading on truck and moving about the mine, making a large saving in time and labor over other designs. By the use of an adjustable friction clutch, through which the feed gearing is operated, all danger of over-

strains and breakages is removed. The power truck is unusually strong and convenient, requiring the movement of one lever to throw the propelling mechanism in or out of clutch.

VENTILATION.

The proper ventilation of deep coal mines is of the utmost importance. Fan blowers of comparatively low speed, or high-speed, low-pressure air compressors are used. For driving fan blowers, either direct-current or alternating-current motors can be successfully used. Where the distribution is by direct current, the motors can be either of the shunt- or the compound-wound type, the latter being preferred for very large fans of which the starting torque is great. With an alternating-current system of distribution, induction motors are often used, their advantage being their high starting torque and the possibility of speed variations for changing the air supply. This latter point, however, is not of great importance, as actual practice has shown that the losses in the rheostatic motor control are about the same as the losses due to a mechanical shuttering of the fan. Where fans can be entirely shuttered, the starting torque is not very large; and considering the rather bad effect that an induction motor has on the power factor of a system, the use of synchronous motors is greatly preferred. Where fans are installed in remote places, induction motors would possibly be more advantageous on account of the little attention they require. If possible, the motors should be direct-connected so as to avoid the use of belting, thus insuring more reliable operation and economy in the required space.

On account of the high speed required by centrifugal compressors, synchronous motors are not so well adapted and induction motors would be preferable for this service. Where a compressor is of large capacity and can be installed in the generating station, it may be driven by an efficient steam turbine, eliminating the generator and motor losses.

BREAKERS AND TIPPLES.

The various auxiliary machines used in breakers and tipples, such as crushers, picking tables, screens, conveyors, etc., are preferably driven by polyphase induction motors, either direct-connected to driving shafts or to individual machines. The absence of commutator troubles, due to the severe vibrations in this service, makes

such motors preferable to direct-current motors. The entire omission, or reduction in the number of belts formerly used in this kind of work naturally results in increased efficiency and consequent decrease in operating expenses, while at the same time it greatly improves the safety and reliability of the installation.

Dredges.

While the early types of steam-driven dredges were partly successful, their operation was rendered expensive by the scarcity of available fuel and the cost of handling it. With the rapid development of hydro-electric plants and large central generating stations, together with the increased distances to which electric current could be economically transmitted, it was found that the electric motor afforded a compact, easily controlled, and highly efficient substitute for steam drive; separate motors could be applied either directly or with short belts to the various units of the dredging machinery, and a larger percentage of the input power was thereby directly applied in useful work.

While many changes were necessary in the type of motor originally applied, before satisfactory results were obtained, motor drive has indisputably proved its superiority to steam drive in cost of both power and maintenance. The modern dredge can now be supplied with motors especially designed for heavy duty, and capable of running continuously without danger, and requiring a minimum of attention.

The standard form of dredge used to-day is the continuous-chain, close-connected bucket type, ranging in capacity from 3 to 13½ cu. ft. The speed of the bucket line varies from 50 ft. (with 18 to 25 buckets) to 75 ft. (with 35 to 50 buckets) per minute, depending upon the condition of the ground. For the operation and control of the bucket line, a variable-speed motor is used. This is located on the lower deck and is belted to a driving pulley, which is generally situated in the rear of the pilot house on the upper deck. The duty imposed upon this motor is severe, as it must operate under conditions calling for power varying from approximately 75 percent. overload down to 25 percent. of its rated capacity.

To keep the dredge in place, move it about, or hold it against the bank when digging, head lines are used, being controlled from the

forward end, and generally operated by a 6-drum winch driven by a variable-speed motor. The winch motor, while of smaller capacity, must be of the same rugged construction as the digger motor. It is equipped with a suitable controller and resistance to permit continuous operation at from one-half to full-load speed. It has been found advisable to equip the motors for this service with solenoid brakes, by means of which the motor can be brought to a standstill almost instantly. It is then ready for the reverse operation without the usual reversing of the motor through the controller. This latter method is bad practice, as the sudden reversing of the motor in this way causes a heavy strain on the windings, and may result in a burnout.

The high- and low-pressure pumps for supplying water to the screens and sluices are generally operated by a separate motor, direct-connected to each pump. The high-pressure pump is operated by a 720- or 900-r.p.m. motor, and the low-pressure pump with a motor operating at 600 to 720 r.p.m.; each motor is mounted on the pump base and is direct-connected to the pump by a flange coupling.

For driving the primary pump, when priming the large pumps, or for supplying water on the table during the clean-up, a 10-h.p., high-speed motor, direct-connected to a centrifugal pump, will insure economy and efficiency in operation, as the motor, when used intermittently, will consume current only in direct proportion to the work done.

For the operation of either shaking or revolving screens, a constant-speed, belted motor of from 25 to 50-h.p. capacity is usually required, and is generally installed on the upper deck. This motor is similar to that operating the winch and the bucket line, but is equipped with a small reversible controller, with sufficient resistance in the armature circuit to bring it to full speed in about one minute.

For driving the conveyor belt of the stacker, a 25 to 50-h.p., constant-speed motor, similar to the one used on the screens, is required, located at the end of the ladder, and either belted or connected by silent-chain drive.

Mine Telephones.

Pre-eminently important from the mine operator's viewpoint, in considering telephone service in a mine, is the saving of time

and the facility with which orders and messages may be orally despatched to employees stationed in various departments of the workings. The savings to be effected by the use of mine telephones are numberless, and may be readily appreciated when considering the variety of characteristic accidents, such as explosions, water freshets and landslides, which make mining a hazardous business. Mine officials frankly say that the presence of this protection is worth the initial cost of the system, which is repeatedly earned by its reliable performances at just such times as those mentioned. The operation of almost every mine is dependent upon a variety of these uncertain conditions, rendering it almost impossible to anticipate a temporary suspension of work in advance of its actual occurrence. The fact that the superintendent of a mine may remain in his office and be in direct talking communication with every corner of the entire mine system at all times is of such paramount importance that it seems unlikely that any mine owner will conscientiously avoid giving this matter serious consideration.

The cost of constructing the underground line for a mine telephone system is usually less than for surface systems, because no poles are required and there are no holes to dig. A mine telephone system does not require an expert to install, and about all the work required of the mine owner is the placing of the instruments to advantage and stringing line-wire to these points. This single feature of construction is quickly done by running wires through the drifts and down the shafts on standard wood brackets equipped with common glass insulators. In a large number of mines, rope haulage is used to convey ore to the surface. Where such is the case, a metallic signal line is usually run along the side of the entry to transmit signals to the engineer, usually located at the surface. If such a condition exists, it is necessary only to connect any required number of bridging telephones to the circuit, provided by the haulage strand and the signal wire. In this case no expense is incurred beyond the first cost of connecting them at each station. The expense of maintenance is usually very light, because lines under ground are not subject to atmospheric conditions that are present outside, and trouble from lightning is, of course, unknown.

Although ordinary, iron line-wire has been used with success in

some mines, the safest construction can be furnished at little additional cost by substituting rubber-covered wire for all underground circuits which are in any way subject to moisture or dampness from contact with fresh earth or with timbers. In some conditions it has been considered good practice to use lead-covered cable; providing the installation warrants the expense. In any event, this part of the equipment should not be slighted, and the best is always the cheapest. By installing cable an increased degree of certainty is assured.

LIGHTING.

The electric system of lighting, for general illumination, is too well known to need any comment. The problem of making a satis-

FIG. 10. ELECTRIC MINERS' LAMP.

factory miner's hat lamp to supersede the present oil lamp, which will always remain a constant source of danger and discomfort, has for many years been the aim of a number of inventors, and numerous designs of more or less value have been put on the market. One type, which has just been developed, promises to be a great improvement in this line.

This lamp consists of a miniature tungsten unit operated from a light, portable, storage battery. It is rated at one mean horizontal candle-power, but by an effective reflector, as high as 5 c.p. is obtained in the beam of light at a distance of 4 or 5 ft. The lamp socket consists of a hard, moulded compound, unaffected by moisture, acid or gases, which completely encloses and protects all metal parts. The steel reflector, which is enameled both inside and outside, is also supported from the lamp socket. The complete lamp is compact, light in weight, and mechanically strong. It is designed to replace the old type of oil lamp, now in general use, without any modification to the cap.

The storage battery is portable, designed to be carried either on a belt or from shoulder straps, or by a handle, as a lantern. The cell is protected by a japanned steel case with an acid-proof moulded cover. The terminals are brought out through an acid- and moisture-proof receptacle, from which an armor-braided, rubber-insulated cable connects to the hat lamp. The battery has a capacity of 10 ampere-hours, and is of sufficient size for operating a lamp 12 to 14 hours. When used as a hand lantern, the lamp socket and reflector are removed from the cap receptacle and inserted into the receptacle on the side of the battery.

PHYSIOLOGICAL EFFECTS OF CARBON MONOXIDE.

BY HENRY S. MUNROE.*

An interesting pamphlet on carbon monoxide has recently been issued by the Bureau of Mines, in which attention is drawn to the dangerous properties of this gas and to the use of mice and birds for detecting its presence in mine air. The author quotes largely from various publications of Dr. J. S. Haldane of Oxford University, who for many years has made special study of the subject of mining hygiene and the dangerous gases met in mines. On page 6 the author states: "According to Haldane, carbon monoxide has no other effect than that resulting from its interference with the oxygen supplied to the tissues, and apart from its property of combining with the hæmoglobin it is physiologically indifferent, like nitrogen." The author also outlines an experiment in which he remained for twenty minutes in an atmosphere containing 0.25 percent. of carbon monoxide, "at the end of which time he suffered only a slight headache, although later he became ill. The illness lasted for several hours and was accompanied by nausea and headache." The quotation from Haldane, and this experiment, are likely to give a false impression as to the dangerous properties of this gas; it has therefore seemed wise to give a few facts to the readers of the QUARTERLY, that others may not be led to repeat the experiment made by the author of the pamphlet, and to give some idea of the dangerous nature of this gas even when present in very small percentage.

Carbon monoxide is a product of incomplete combustion. It is present in large quantities in producer gas and water-gas, and in dangerous amounts in the gases from boilers and furnaces of all kinds. It is often present in large proportions, and always in dangerous amounts, in powder smoke, in the gases from underground as well as surface fires, and in the after-damp from explosions of firedamp and coal dust.

Carbon monoxide has the property of forming a compound with

* Professor of Mining, Columbia University, New York.

the hæmoglobin of the blood. The effect of this is to make the hæmoglobin, so combined, practically inert and to prevent it from acting as a carrier of oxygen. When so much carbon monoxide is absorbed that the greater part of the hæmoglobin is inert, death results. The affinity of carbon monoxide for hæmoglobin is over two hundred times greater than that of oxygen, so that when present in the air, even in small quantities, it is freely absorbed by the blood. Carbon monoxide is not displaced by oxygen but is dissociated by natural processes, and escapes in the expired air. Where large quantities are absorbed, it may be several days before the last traces disappear. According to Drs. Edsall, v. Jaksch, Haldane, and other authorities, 0.05 percent. of carbon monoxide is dangerous. According to Haldane, severe symptoms were observed from breathing air containing 0.02 percent., or one part in five thousand. With this small amount present the blood becomes 20-percent. saturated after about 20 hours, producing slight giddiness and shortness of breath. At this point an equilibrium seems to be established, and the dissociation of the gas keeps pace with its absorption. With increasing percentages of carbon monoxide, the saturation of the blood becomes greater and the time required to produce the maximum effect shorter. With 0.08 percent. present, the blood becomes 50-percent. saturated within a few hours; it becomes scarcely possible to stand and even slight exertion results in loss of consciousness, the senses are confused and the judgment is impaired. Sometimes the victim either becomes stupid and drowsy, or much excitement results, not unlike the effects of alcohol. Another experiment by Dr. Haldane proved that with 0.20 percent. CO in the air, the blood becomes 50-percent. saturated in 70 min. With 0.25 percent., the amount present in the Bureau of Mines experiment, this dangerous condition would be reached in less than one hour.

According to v. Jaksch, the absorption of 0.8 gram of carbon monoxide is fatal. According to Haldane, if death occurs gradually the hæmoglobin is usually about 80-percent. saturated with carbon monoxide. Post-mortem examinations of persons who have died from carbon monoxide poisoning show that the effect is to produce intense congestion of the vital organs, especially in the brain, usually accompanied by small hemorrhages. It is possible

that this congestion is due to the attempt of nature to make good the diminished efficiency of the blood by supplying larger volumes at needed points.

Even when death does not occur, very serious results are likely to follow from the absorption of this gas by the blood. The after effects are lesions, cysts and local softening of the brain tissue,* inflammation of the membranes of the stomach and intestines, pneumonia, bronchitis, pleural effusions, inflammation of the kidneys, fatty changes in the heart, ænemia, splenic enlargement and other derangements of vital organs, sometimes resulting in death even after several years. It is believed that Sir Clement LeNeve Foster was a victim to carbon monoxide poisoning which occurred on a visit as chief inspector of mines to a mine in Cornwall some years before his death. From the very full record given by Mr. Foster of his symptons while exposed to the gas underground it does not appear that there could have been more than 0.08 percent. of carbon monoxide present, nor that his blood could have been more than 50-percent. saturated, although direct evidence on both these points is lacking. The experiment made by the author of the paper recently issued by the Bureau of Mines, in which he exposed himself for twenty minutes to an atmosphere containing five times as much carbon monoxide as is known to be dangerous, was therefore hazardous and even though the experimenter apparently suffered but little ill effect a somewhat longer exposure would certainly have resulted in serious injuries, the after effects of which might have proved fatal.

One of the most serious dangers from the presence of carbon monoxide in the air of mines is the effect upon the health of workmen who are daily exposed to the breathing of small amounts of this gas. The blood, when partly saturated, is thereby rendered less able to perform its proper functions, so that the patient suffers

* In a personal letter, Professor Walter B. James, of the College of Physicians and Surgeons, states that a peculiar local lesion of the brain, with softening of the lenticular nucleus is fairly characteristic of carbon monoxide poisoning. The action of the gas upon this area has not been satisfactorily explained. It has been suggested that it is due to the peculiar angle at which minor blood vessels are given off to this area from the arterial trunks.

According to Dr. James, the remote consequences of monoxide poisoning are a serious matter. His cases have shown mental aberration of a peculiar type, with great slowness to response, going on to cerebral degeneration and death.

from ænemia and all the complications that may result from this weakened condition. According to Dr. Edsall, the disease known as miners' phthisis has been shown to be due chiefly to carbon monoxide poisoning. Recent observations have shown that for some hours after a blast, under ordinary mining conditions, carbon monoxide may be present in the air in dangerous amounts, and undoubtedly men engaged in sinking, drifting, and stoping where the circulation of air is deficient have their blood partially saturated with carbon monoxide the greater part of the time.

By some authorities it is believed that the serious effects above outlined, due to absorption of carbon monoxide by the blood, are supplemented by direct toxic action on the nervous system, on the muscles, the heart and other organs. It is believed by others that there is a cumulative action and that those who have been poisoned by this gas are more likely to become victims when again exposed to it. It is quite certain that dissociation of carbon monoxide from the blood is slow and that those whose blood is partly saturated will sooner fall victims where larger quantities of the gas are breathed than those whose blood is free from this gas. Men who have repeatedly suffered from carbon monoxide poisoning become very sensitive to the gas, and in most instances are compelled to abandon work in which they are compelled to breathe air containing it.

The symptoms by which carbon monoxide poisoning may be detected are not difficult of recognition. The blood becomes a brilliant cherry red, and in serious cases red or bluish-red spots appear on the front of the neck, on the trunk, thighs and elsewhere, lasting for some days, and in fatal cases apparent after death. The mental disturbances, weakness and lassitude, have been noted. This is followed by headache, accompanied by nausea, often lasting 24 or 48 hours, even in slight cases. In more serious cases, the headache may recur at intervals for some months. Loss of consciousness with convulsions, may occur several hours after the poisoning. One of the first symptons is weakness in the knees and legs, sometimes lasting for days, with aching from the knees to the ankles. Local pains in the region of the heart, and palpitation of the heart, are common and may recur at intervals for a month or more. Foster, and several others, have published valuable notes

on these symptons, which will be found in the appendix of Foster and Haldane's Investigation of Mine Air.

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STATIC AND DYNAMIC FORCE RELATIONS IN MACHINES.

BY WALTER RAUTENSTRAUCH.*

Every machine is composed of stationary and moving parts which serve to transform one kind of motion into another kind of motion, and to transfer forces applied to one part, moving in a given path, to other parts, moving in the same or different paths. In slow moving machinery, such as brick presses, toggle-joint riveters, hydraulic presses and the like, in which, although the masses of the parts may be great, their rate of change of velocity is at no time large, the forces acting on the several machine parts, and the reactions of their supports, are due solely to the structural characteristics of the mechanism. In high-speed machinery, such as steam and gas engines, wood working machines, printing presses, spinning and weaving machines and the like, the kinematic characteristics materially influence the distribution of the forces and their resulting reactions throughout the machine. The accelerating and retarding of large masses at even low rates, or of small masses at high rates, result in the storing up and giving out of energy during certain parts of the cycle at such rates as to occasion a far different force application than the purely structural characteristics would indicate. However, the static and dynamic force relations in machines, consequent to the structural and kinematic characteristics of the mechanisms, admit of correlation and analysis by simpler methods than first thought would lead one to suspect.

It will be well first to realize that the application of force to a machine, such as a given belt pull at a pulley rim or a tooth pressure at the pitch circle of a gear, is possible only when an equal and opposite reaction is offered by the machinery. The reaction may be the resistance of a dynamo to turning, or the pressure on a tool, or the weight and inertia of a column of water, each of which may act in combination with the inertia of the moving masses of the machine itself. The extent to which each factor may contribute to the resistance or reaction at any phase of the cycle may vary,

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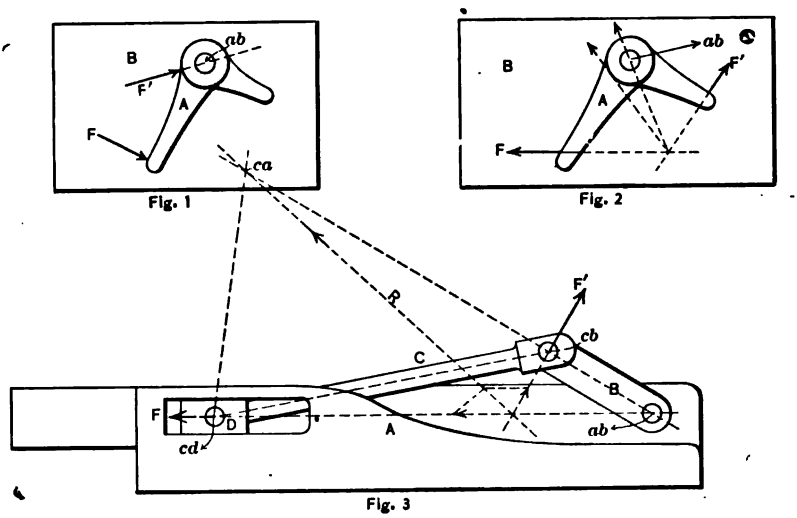
but the fact remains that the sum of all resistances must at any instant give a reaction at the point of application of the driving force equal to the force itself. When a constant driving force is applied to a machine at any point, a speeding or accelerating of the moving masses will result when the force against which the machine is operating falls off, and a deceleration or retardation of the machine parts is experienced when the forces resisting the machine movement are increased. At all instants, however, driving force is equal to static resisting forces plus or minus the mass times acceleration of the machine members (reduced to the point of application of the driving force). This relation being true, it will at once be clear that the principles of mechanics relating to static equilibrium of structures and mechanicisms may be applied for any phase relation of the machine.

Instant Centers and Static Force Relations.

When a lever such as A is constrained to rotate about B at a point ab, as shown in Fig. 1, it is evident that rotation can take place only upon the application of a force, such as F, acting with a leverage about the point ab. Should there be only a force F' acting on A in a direction passing through the center of rotation ab, there will be no turning moment on A and consequently no motion of A with respect to B. Should there be a force F acting on A with a given leverage about ab, and at the same time a force F', as shown in Fig. 2, acting with a given leverage also about ab, and should the resultant of F and F' pass at a distance from ab, there would be a tendency for A to turn with respect to B, which would be proportional to the product of the resultant of F and F' and the distance of its path from ab. This torque will be equal to the moment of inertia of A about ab times the angular acceleration of A. When this distance becomes zero, that is when the resultant passes through the center ab, there will be no tendency for A to turn with respect to B, and they will remain at rest with respect to each other. It will appear, therefore, that in general when the resultant of all forces acting on a body passes through the point of rotation of that body with respect to any other body, there will be no tendency to rotation of the one body with respect to the other.

If then in any machine the centers of rotation of the members,

one with respect to another, are established for any phase relation, the amount and direction of the forces or reactions in all members resulting from the application of a given force in a given direction in any member can be readily determined for that phase. First, we must know the centers of rotation of members, one with



FIGS. 1-2-3.

respect to the other. To accomplish this we need only to know how two points in any member are constrained to move when the other member, with respect to which the motion is to be defined, is stationary.

Thus, consider the case of the engine mechanism in Fig. 3, in which it is required to know the motion of C with respect to A for the particular phase shown. It is evident from the figure that the crank-pin end of C is constrained to move, at the instant, in the direction indicated by the arrow, while the cross-head end of C is constrained to move in the path of the engine center-line. Hence, normals to the direction of motion at the points in question locate by their intersection *ca* the center about which C moves with respect to A at the instant. Therefore *ca* is the instant center of C with respect to A at the particular phase shown. A force F applied to the cross-head in the direction of the engine center-line will

move the connecting rod C with a uniform velocity, or without appreciable acceleration, when the reaction at the crank-pin F' is such that the resultant (of F and $F' = R$) passes through ca. Should the force F be a given amount fixed by gas pressure, and

FIG. 4. TOGGLE-JOINT RIVETER.

the reaction F', fixed by the load of the engine, be such that a resultant is obtained which does not pass through the instant center ca, an acceleration of C will result, which will be proportional to the torque of the resultant about ac and the moment of inertia of the rod mass about the same point. Hence, having given a known force F acting at the cross-head pin in the center-line of the engine, the reaction at the crank-pin in the direction of the tangent to the crank-pin circle, necessary to afford a uniform velocity of the rod C, will be obtained by the parallelogram of forces, as shown in Fig. 3. In machines in which the acceleration of the parts is of no consequence, the above methods may be employed to trace the force relations throughout the entire machine. As an illustration of the application of this method, the following example and its solution are given.

Let it be required to determine the force which may be applied to a rivet by means of the machine shown in Fig. 4. Also, with a constant pressure in the cylinder (air or water), how does the force on the rivet vary throughout the stroke of the plunger? Again, for any position of the mechanism, what is the force along *D* (for example) when a given pressure is maintained in the cylinder? The solution of the first two problems relates to the characteristics of the machine; the last problem to the proportioning of the parts.

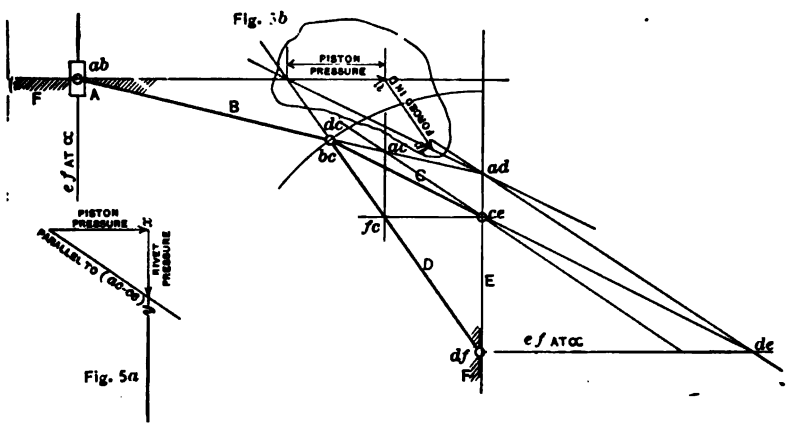


FIG. 5.

Two positions of the mechanism will be taken as sufficient to illustrate the problem. Let Figs. 5 and 6 show diagramatically the position of the links for two piston positions. For each phase, let the instant centers, *ae* and *ad*, be found. (In this solution, Kennedy's theorem that the instant centers of three bodies constrained to have relative motion lie in the same straight line, has been employed.) In each case it will be observed that *ae* lies in the line joining *ac* and *ce* and on the line joining *ad* and *de*, which lines are parallel; hence *ae* lies at infinity in direction of these lines. Accordingly, then, as shown in Figs. 5a and 6a, if we lay off the force behind the piston *A* to scale, and represent its amount and the

direction of its application, and from one extremity of this force line draw a line in the direction of the force at E on the rivet, and from the other extremity a line in the direction of $(ac\text{-}ce)$ or $(ad\text{-}dc)$, that is, to ae, these lines will define by their intersection the force at E in the direction of pressure on the rivet. The force on the rivet in each case is xy. For the position of the mechanism shown in Fig. 5, the force in the link D, in the direction of D, is found, as shown in Fig. 5b, by laying off the force behind the piston in proper direction and amount, from one extremity drawing a line parallel to the link D, and from the other extremity a line to the instant center ad. The length $w\text{-}v$ in the diagram, Fig. 5b, will represent to scale the force in the link D along D.

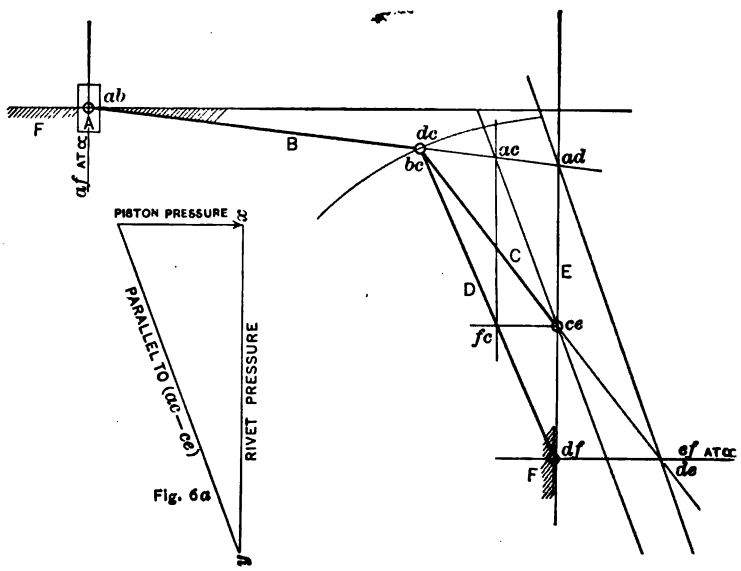


FIG. 6.

In like manner, any machine, no matter how complicated the mechanism, when once the instant centers have been determined, can be analysed for the static force relations between any pair of members.

Dynamic Force Relations.

For machines in which the moving of masses occasions important inertia forces, a most convenient study of force relations is accomplished by reducing all forces acting in the machine to some single prominent point in a moving member, and reducing the masses to that point. Such a point in an engine is the crank-pin center. This method of analysis was first suggested by Prof. Wittenbauer in an article in *Zeitschrift für Mathematik und Physik*, Vol. 50, and was applied to the determination of flywheel weight of an engine in an article in the *Zeitschrift des Vereines deutscher Ingenieure*, Vol. 49.

To accomplish this force and mass reduction, two fundamental principles in the theory of machines are employed. First, any force P applied at a given point on a member of a machine moving with a velocity v in a given path, will cause a reaction P_n at some point N in another member of the machine constrained to move with a velocity v_n in a given path. Thus, in general:

$$P = \sum \left[P_n \frac{v_n}{v} \cos \theta \right]$$

where θ is the angle between P_n and v_n. This equation states analytically what previously has been set forth graphically. Second, any mass M of a machine part whose mass center is moving with a velocity v has a capacity for doing work, or energy, equal to $\frac{1}{2}Mv^2$. It will give up its energy to another mass M_m located at some mass center N, and give it a velocity v_n such that

$$\tfrac{1}{2} M_v^2 = \tfrac{1}{2} M_n v_n^2.$$

Thus we may write in general,

$$M = \sum \left[M_n \left(\frac{V_n}{V} \right)^2 \right]$$

It will be observed that any mechanism or machine may be reduced to a moving point of variable mass acted upon by a variable force. In reducing the masses of members moving in complex paths to equivalent mass at the reduction point, it is found con-

venient first to split up the total mass of the member into substitute masses concentrated at points in the member, the paths of

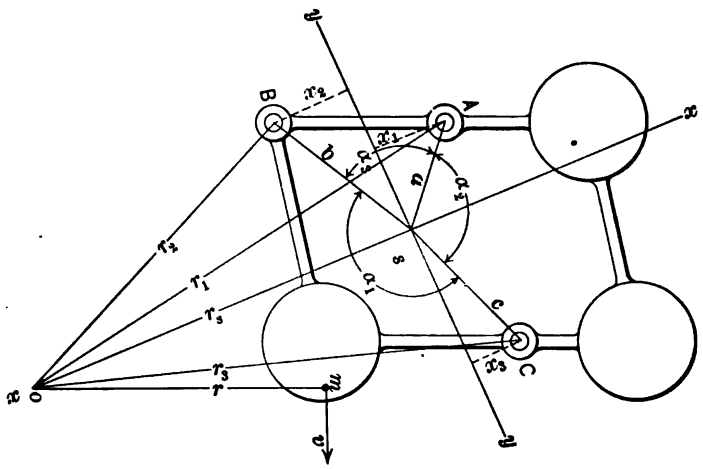


FIG. 7.

which points are more regularly defined. This may be accomplished as follows:

Let Fig. 7 represent a machine member of mass M constrained to move in a complex path, but three points of which move in simple paths.

Let S = center of gravity of the member.

O = its instant center in respect to the frame of the machine.

m = any mass point.

v = velocity of m.

E = kinetic energy at any instant.

Then $E = \sum(\frac{1}{2} mv^2) = \frac{1}{2} I_o w^2$

where I_o = moment of inertia of M about O.

w = angular velocity of M about O.

Let A, B, C, and S be the points to which the mass is to be

reduced, such that the energy remains the same, which means that:

1.—Their sum must equal M.

2.—Their center of gravity must be at S.

3.—Their moment of inertia about S is equal to that of M.

We may then write:

$$M_a + M_b + M_c + M_s = M$$
$$M_a x_1 + M_b x_2 + M_c x_3 = 0$$
$$M_a y_1 + M_b y_2 + M_c y_3 = 0$$
$$M_a a^2 + M_b b^2 + M_c c^2 = Mk^2 = I_s$$

The solution of the four equations results in:

$$M_a = M\frac{k^2 \sin a_1}{a\,l} \qquad M_b = M\frac{k^2 \sin a_2}{b\,l}$$
$$M_c = M\frac{k^2 \sin a_3}{c\,l} \qquad M_s = M - \left(M_a + M_b + M_c\right)$$

where $l = a \sin a_1 + b \sin a_2 + c \sin a_3$

To prove that the kinetic energy is unaltered when for M the masses M_a, M_b, M_c, and M_s, are substituted at the points A, B, C, and S, respectively, let

E_1 = kinetic energy of the four masses.

$$= \frac{w^2}{2}\left[M_a r_1^2 + M_b r_2^2 + M_c r_3^2 + M_s r_3^2\right]$$
$$= \frac{w^2}{2}\left[M_a\left(r_1^2 - r_s^2\right) + M_b\left(r_2^2 - r_s^2\right) + M_c\left(r_3^2 - r_s^2\right) + M r_s^2\right]$$

From Fig. 7 it is seen that:

$$r_1^2 = r_s^2 + a^2 - 2\,r_s x_1$$
$$r_2^2 = r_s^2 + b^2 - 2\,r_s x_2$$
$$r_3^2 = r_s^2 + c^2 - 2\,r_s x_3$$

Therefore:

$$E_1 = \frac{w^2}{2}\Big[M_a a^2 + M_b b^2 + M_c c^2 + M_s r_s^2 - 2 r_s\Big(M_a x_1 + M_b x_2 + M_c x_3 \Big)\Big]$$

$$= \frac{w^2}{2} M\Big(k^2 + r_s^2 \Big) = \frac{w^2}{2} I_0$$

Hence $E_1 = E$.

Following Wittenbauer's analysis, let it be required to determine the behavior of an engine having known dimensions, and weight of parts, operating under a known steam pressure variation, and against a known and constant resistance. Let Fig. 8 represent

FIG. 8.

such an engine. The connecting rod, the crank and the flywheel in this case are the members in which mass reduction is required. Referring to the connecting rod, it is found that the total mass of the rod may be reduced to the two ends and the center of gravity.

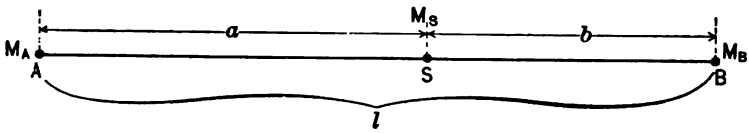


FIG. 9.

Let Fig. 9 represent the rod diagramatically, A being the cross-head end, and B the crank-pin end.

$$M_a = M\frac{k^2}{al}$$

$$M_b = M\frac{k^2}{bl}$$

$$M_s = M - \left(M_a + M_b\right)$$

where k = radius of gyration of rod about an axis through S perpendicular to the plane of motion.

Let the crank-pin center be the point to which all moving masses are to be reduced. These masses will be:

1.—The reciprocating parts $+ M_a = M_1$

2.—The rotating parts $+ M_b = M_b + M_2$

3.—M_s

Concerning M_1 and M_s, it will be required to have the relative velocities of M_1, M_s, and the crank pin. This is accomplished by the following well-known construction:

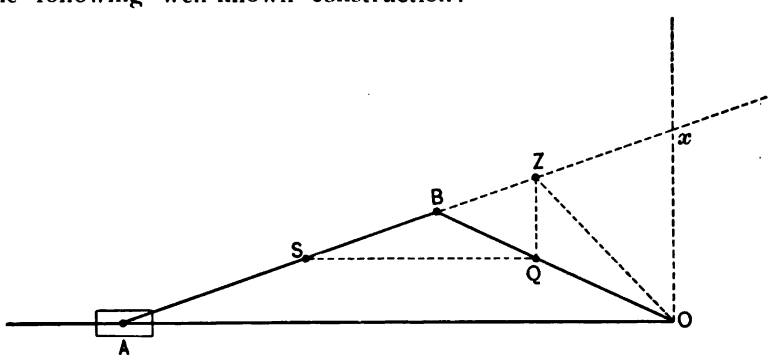


FIG. 10.

Let Fig. 10 represent one position of the mechanism; then if AB

be extended to cut a vertical through O (OA being horizontal), OX will equal the velocity of A to the same scale on which OB represents the velocity of the crank pin. Also let SQ be parallel to OA, and QZ be perpendicular to SQ.

Then $$\frac{OZ}{OB} = \frac{\text{velocity of } S}{\text{velocity of crank-pin}}$$

The velocity of all rotating masses with respect to the crank-pin will be directly proportional to their mass-center radii about the center of rotation (shaft center).

The obtaining of these velocity relations for a number of positions of the mechanism will permit the determination of the equivalent mass at the crank pin for all positions in its path, by means of the equation previously set forth. The crank-pin circle may then be rectified and laid out as TR in Fig. 11, and ordinates representing reduced mass be laid out for each corresponding crank position, giving the curve G-G in the figure. The indicator cards for the engine may next be laid out, and the total force behind the piston at any instant may be transferred to tangential force at the crank-pin by the methods already explained. From each of these values of tangential force must be subtracted the constant

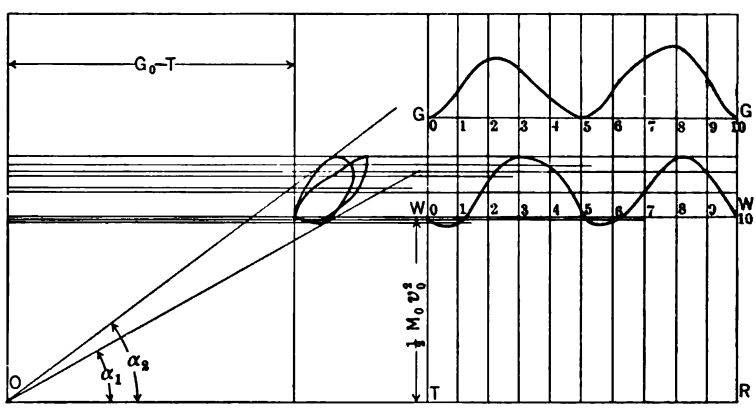


FIG. 11.

force at the crank-pin caused by the engine's working against a uniform torque. (For any other conditions of load, these resistance forces at the crank-pin may be variable.) There will then

remain the force which is effective in accelerating the mass reduced to the crank-pin. These forces, corresponding to given crank positions, may then be laid out with the rectified crank-pin circle as a base. The integration of the resulting curve up to any crank position will give the work done up to the crank position in question, in accelerating the masses. This work diagram will then appear as *W-W* in Fig. 11, and is laid out on a base line which lies above *T-R* by an amount, to scale, equal to the energy of the machine at the beginning of the stroke, that is, $\frac{1}{2} M_0 v_0^2$, where M_0 is the mass reduced to crank-pin at the beginning of the stroke, and v_0 is the velocity of the crank-pin at the beginning of the stroke; v_0 is found from the mean speed at which the engine is to run.

If now the work done up to any crank position is taken as ordinates, and the reduced mass at the crank-pin is taken as abscissas, a diagram such as shown to the left in Fig. 11 results. This diagram, which may be termed the energy-mass diagram, possesses some valuable properties in studying the characteristics of the machine under the conditions of load and powering assumed. It will be noted that the origin of the energy-mass diagram is at *O* and that a tangent from *O* to the diagram will represent the relation:

$$\frac{\text{Work done}}{\text{Mass}} = \frac{\frac{1}{2} Mv^2}{M} = \frac{1}{2} v^2$$

and will therefore be a measure of the velocity of the machine at that particular crank-pin position through which the tangent passes in the diagram. Drawing the tangents to the boundaries of the energy-mass diagram, we obtain a_1 and a_2 as a measure of the minimum and maximum limits of velocity which the crank-pin will attain during the cycle, and the points of tangency will define the crank-pin positions at which these velocities will occur.

While the illustration given for the application of the above principles has referred to a particular type of machine, the relations hold for any type whatsoever, and it is a most accurate method for determining the dynamic relations in machines driven by a known system of force application and working against a known system of resistances.

COLUMBIA UNIVERSITY, April, 1912.

CAISSON FOUNDATIONS FOR THE SEAMEN'S CHURCH INSTITUTE BUILDING.

BY MAURICE DEUTSCH.*

This building, now in course of construction, is situated on a lot 91x112 ft. at the northwest corner of Cœnties Slip and South Street, overlooking the East River at a distance from it of about 70 ft. It is to be 12 stories above the curb and three floors below the curb, of the same area. The lowest floor is 31 ft. below the curb or 27 ft. below high water in the river, and will contain boiler, engine room and general storage. The sub-basement floor is 19 ft. below curb and will contain a refrigerating plant, coal storage, etc. The basement floor is 5 ft. below curb and will contain a restaurant and offices.

Ground Conditions.

The site was originally a part of the East River, and resembled in appearance the mud flats now seen along the shores of the Harlem River at low tide. As no excavations had yet been carried to bed-rock in this vicinity, a number of carefully driven borings were taken in which great difficulty was encountered in penetrating the material overlying bed-rock. The depth to rock was found to vary from 30 to 35 ft. below curb, and the slope appeared to be toward the East River. In order to ascertain that bed-rock and not boulders were encountered at these depths, three borings were taken at the extreme ends of the lot, drilling a distance of 5 ft. into the rock by diamond drill. The specimens of core indicated a very good mica schist. The average depth to rock on the entire lot, as actually found in the caissons, was 37 ft. below curb. The rock was found to be fairly level, unusually so for lower Manhattan; at 80 Maiden Lane, for example, a variation of 10 ft. in rock surface was found within several caissons.

To provide the three floors below curb would have been practically impossible by any other method than by the use of pneumatic caissons, forming a water-tight cofferdam around the perimeter of

* Consulting Engineer, 50 Church St, New York. C. E., Columbia, 1906.

the lot and bonded with water-tight joints to the bed-rock. The level of the ground-water varied with the tide in the East River, and at one time was observed to be within 3½ ft. of the curb. The borings also showed that stone-filled cribs and other varied timber would be encountered. In view of this condition, and the heavy column loads, a pile foundation was practically impossible.

Specifications and Contract.

The minimum width of caisson was specified, and also that all timbers in the caissons exposed during the excavation of the interior of the lot were to be easily removable. The roof of the working chamber in the caissons was to be sloped from the center to each side, so as to overcome difficulties heretofore found in obtaining a tight bond between the concrete in the working chamber and that used to fill in the shafts when the caisson reached bed-rock, eliminating thereby any possible chance of void spaces. The character of caissons to be used by the contractor was not otherwise defined, but he was, however, required to provide working drawings which were to be submitted to the architect and the engineer for approval. This gave the contractor an opportunity to adopt such details of construction and methods as were best adapted to his plant and experience.

It was expected that considerable difficulty would be experienced in maintaining caissons perfectly vertical during the sinking, but it was required that in case any caisson were displaced more than 6 in. from the vertical, the contractor was to provide a remedy at his expense. The new city regulations, prohibiting the projection of any part of the foundations beyond the building line made it necessary that the caissons supporting the outside columns of the building be eccentrically loaded. Owing to these conditions, the contractor asked permission to start the caisson sinking with a clearance of 4 ft. from the building line, in order to allow for possible divergence caused by obstructions during sinking; this was granted with the proviso that the caissons should finally be landed within the required limits. Only one caisson was as much as 6 in. out of plumb. Such deviation from the vertical was a matter of extreme importance, as it would tend to increase the eccentricity, or tilting effect, of the column loading.

The space between the roof of the working chamber and the

concrete filling placed in the latter, after the caisson reached rock, was to be grouted under air pressure. All caissons were required to be sunk to bed-rock and the surface of the rock in the working chamber was to be made level and clear of fractured or unsound portions. The contractor was required to do all the pumping from the interior of the lot until his work was completed, so that practically no expense for this item had to be figured by the waterproofing contractor. The contractor assumed all responsibility for damage done to adjoining property. The contractor was required to furnish a lump sum for carrying the caissons to a certain assumed average depth below mean high-water datum, and an extra price per cubic yard for work below that depth. This method of placing a contract for foundations commends itself as being fair to both owner and contractor.

Caisson Design and Construction Details.

The caissons employed on this work were practically of the same design as the reinforced concrete caissons used on the Municipal and Woolworth buildings. The caissons were entirely of reinforced concrete; they had a uniform width of 5½ ft. and lengths varying from 20 ft. 8 in. to 24½ ft. with the exception of one special caisson, designed to contain a single opening for an ashlift. This latter caisson, 13¼x14 ft., was located at the extreme southwest corner, on Cœnties Slip, and was the only one which projected at all beyond the building line. The walls of this caisson were made 3 ft. thick and when it had reached bed-rock, an opening about 5 ft. high was cut through the wall at the bottom, to give access from the cellar floor to the ashlift.

The Z-shaped cutting edge was made of a 9-in. horizontal plate and two flange angles, and had holes to receive the 1-in. reinforcement rods, which had double nuts engaging it on both sides. One set of rods was parallel to the outer surface of the caisson, and passed through holes in horizontal 3x4-in. reinforcement angles about 3 ft. apart, with their vertical flanges in the exterior face of the concrete. The other set of rods was vertical at the upper end, but at the lower end was bent parallel to the sloping inner face of the wall of the working chamber, serving as a framework for the 2-in. vertical boards in the wall of the form. Horizontal ¾-in. rods, 9 in. apart, were wired to the vertical rods, and

the forms on the inner face of the working chamber were wired to the horizontal angles in the exterior walls. The long sides of the caissons were braced transversely by T-shaped horizontal struts made of pairs of angles, back to back, and spaced 4 ft. apart along the cutting edge.

The Sinking Process.

When the foundation contractor commenced on Sept. 1, 1911, to excavate the interior of the lot to about water level, before placing the caissons, he found numerous piles, the remains of old foundations, piers, cribwork, etc., which had to be pulled before caisson work was begun. Many of these piles, as well as those encountered inside the caissons during sinking, were 14 in. diameter and were in a remarkable state of preservation, though probably buried there over 100 years. Considerable difficulty was encountered in the sinking of some of the caissons near South Street, the cutting edge being held up by broken piles and sunken timber. Two of these caissons had to penetrate a heavy crib 24 ft. high, filled with rip-rap. It was made of round timbers drift-bolted together. The bottom of the crib was a solid course of logs and the sides were made of pairs of logs separated by short vertical timbers. The logs, of white pine, were cut with steel chisels 2½ to 3 in. wide. An old scow of heavy oak was also encountered.

Practically all of the caisson forms were built in the contractor's yard and when delivered to the site were immediately placed in position for building up the working chamber and the upper part of the caisson. Concrete was first built up from the ground level to a height of 12 ft. above the cutting edge, or 7 ft. above the top of the working chamber. As soon as this concrete had hardened, the ground inside of the working chamber was excavated and the caisson 'ditched' down as deep as possible, in some cases, as much as 6 ft. below mean low-water level, advantage being taken of the 4 ft. variation in the ground-water level, due to the tide. This much penetration into the mud gave a certain stability to the caisson at the start, before compressed air was admitted. The remainder of the concrete for the caisson was then built up to the full height, as estimated from the borings; it was intended that this height should be such that when the caisson reached rock

the upper end of it would project just above ground-water level. After the air-lock was put in position, compressed air was admitted to the working chamber and the excavation and sinking started. The maximum pressure required on this work did not exceed 18 lb. per sq. in. The caissons passed through 10 ft. of soft, wet mud; 10 ft. of stiff blue clay; 5 ft. of fine sand; 1 ft. of coarse sand; and from 1 to 9 ft. of hardpan, before bed-rock was reached. The thickness of the hardpan was comparatively uniform, and it contained unusually few boulders. The upper layer of soft mud contained interesting relics from the Dutch and revolutionary periods. Sinking was carried on in the usual manner with pick and shovel.. The first caisson was put under air pressure on September 2 and the pressure was removed from the last caisson on October 16, 1911.

Bonding of Adjacent Caissons.

The caissons were designed not only to carry the actual column loads and to resist external pressures, but also to exclude absolutely all ground water from the interior. To accomplish this, it was necessary to secure a tight bond between the ends of adjoining caissons. Spaces of about 2 ft. were left between adjacent ends or sides of caissons and recesses were cored in the opposite faces of the concrete. After the caissons were sunk, wide sheet piles, made of thick planks bolted together, were driven across the spaces between the caissons, with their edges in contact with the faces of the concrete. The sheet piles were driven by a 500-lb. drop-hammer to a depth of about 14 ft., below the surface of the ground; the mud between them was then excavated while the water that seeped into the workings was kept down by pumping.

A section of air shaft was then set in the upper part of the space and concreted to the caisson; an air-lock was attached to the top of it, pressure was admitted, and the shaft was excavated nearly down to the top of the working chamber. The planks removed from the recesses in the ends of the main caissons were turned 90 deg. and spiked to the vertical corner pieces, in recesses, to retain the earth at the sides of the excavation. The sheeting was puddled with clay and the excavation was usually carried to the bottom of the recesses, below which short lengths

of sheeting were driven, engaging slots cast in the ends of the caissons and enabling the excavation to be continued down to the cutting edge; after which it was concreted in the usual manner, thus forming a key providing continuous concrete, and bonding the caissons together around the entire circumference of the lot.

Open Excavation and Cross Bracing of Caissons.

After the caissons had been sunk to bed-rock and the adjacent ends bonded as above described, a complete water-tight box then surrounded the lot. It was then possible to excavate the interior to a depth of 30 ft. below curb, or 27 ft. below water level, without the use of compressed air. The small amount of water that entered the lot was drained by a single 5-in. Worthington steam pump, working only part of the time. This interior excavation, consisting of about 160 cu. yds. was completed within 20 days. The excavation for the interior piers was then carried through the hardpan to solid rock.

To prevent the caissons from being forced inwards by outside pressure, it was necessary during this excavation to brace them across the lot. This was done by placing an upper tier of 12x12-in. braces about 6 ft. below the top of the caissons and 12 ft. apart on centers, both longitudinally and transversely. The excavation was carried deeper and the second tier was set in position about 7 ft. below and supporting the upper tier by vertical shores at intersections. A third tier of braces was then placed a few feet above the hardpan. The precaution was taken so to place these braces across that they would clear all columns, beams and girders, as it was expected that they would remain in position until the steel work was erected above the street level and the girders and beams for the basement floors were set in position and bearing against the caissons to transmit their horizontal thrust across the building. In addition to this, the bracing system was stiffened by placing 3x12-in. planks between the upper and lower tiers, forming trusses.

Underpinning Adjoining Buildings.

The lot is bounded on two sides by old brick buildings with 16-in. walls on pile foundations with timber caps at about low-

water level. As the sinking of the wall cassions in the soft mud almost in contact with these foundations was certain to cause a bleeding of the earth, likely to produce injurious settlement, provision was made to compensate the latter before caisson sinking was commenced. Holes about 8 ft. apart were cut through the cellar walls adjacent to the Seamen's Church Institute and on the opposite sides of the buildings; in them were set groups of three 24-in. I-beam needles 24 to 28 ft. long, which took bearing on their upper flanges against both walls. These served as cantilevers fulcrumed, about one-fourth of the distance from the Seamen's lot wall, on sets of six 3-in. jack screws. Cribbing of 12x12-in. timbers distributed the pressure over about half the cellar area. Accurate levels were taken and recorded, inspections were frequently made, and the jack screws were adjusted as often as any perceptible displacement was observed. Thus the building was maintained at the original level until after the completion of the caisson work, when the walls below the needlebeams were removed. New concrete piers, 4 ft. long and 18 in. thick, were built in pairs, one on each side of the group of needles; these were capped with pairs of 18x18-in. wedging stones, between which pairs of steel wedges were driven and grouted in the usual manner to transfer the weight of the wall from the needles to the new piers, which were forced down on the old footings. The pile caps were thus finally established below their original positions and the foundations are considered to be improved rather than injured by the change. New concrete footings between the wedging piers were built on the pile caps up to within 2 ft. of the old walls, and the space between them closed and wedged with new brickwork in the same way as the piers. No difficulty was experienced in carrying the old brick walls between the needlebeams, as the pressure immediately developed arch action sufficient to span the distance. Horizontal timbers were laid on the bottom flanges of the ends of the needlebeams to support the floor and relieve the lower part of the wall from its weight.

Messrs. Warren & Wetmore were the architects of the building. The sub-structure above described was designed and its construction supervised by the writer. The general contract was awarded to John Downey and the work described was executed by The Foundation Company.

THE PARKS ELECTRO-CYANIDE PROCESS.

BY JOHN R. PARKS.*

Heretofore, practically all successful metallurgical processes in which cyanide salts have been used to extract gold and silver from their ores have contemplated:

1. Removal of coarse particles of the precious metals by amalgamation in a separate device before the ore is submitted to cyanide attack.

2. Presentation of the more or less finely crushed ore to chemical attack by an aqueous solution of cyanide salt, to which is added an alkali to neutralize acid radicals of the ore, to save cyanide salt; this solution must also contain oxygen, either the normal amount that water will dissolve from the atmosphere, or a superabundant amount added by aero-agitating devices.

3. Separation in a clear state, by settling and filtering devices, of the aqueous solution containing gold and silver cyanides, unused cyanide salt, and complex impurities, cyanates, sulphocyanates, etc., of the base metals.

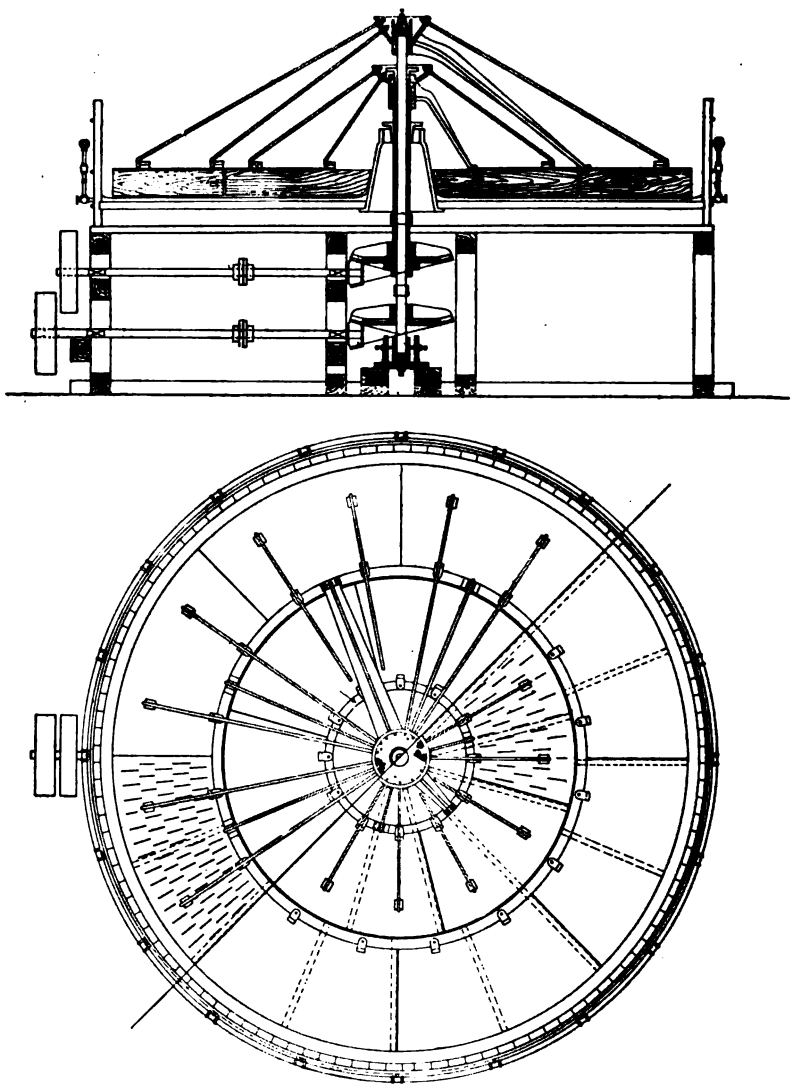
4. Continued and perfect contact of the cyanide solution with the shavings or dust of an inexpensive base metal which precipitates part or all of the precious metals.

5. Separation of the precipitated precious metals from the remaining base metal, washing, collecting, drying and compressing the finely divided precious metals for melting and casting into bullion.

The object of the Parks electro-cyanide process is to apply economically all of the chemical and physical forces necessary to extract and collect the precious metals, applying them at their highest potency, and simultaneously. This necessitates economy in chemicals, their almost instant presentation under their best operating conditions to every particle of precious metal in the ore, and for only such length of time as is required for a thorough chemical interchange. That this has been accomplished will be shown by quoting results of its application to simple and to refactory ores;

* Hollywood Station, Los Angeles, Calif. E. M., Columbia, 1880.

all of these ores, in their raw state, were made to yield their precious metals rapidly in the form of amalgam, in a single operation, in a single machine, eliminating all filtering and precipitating devices.



PARKS' ELECTRO-CYANIDE PAN.

The Parks electro-cyanide pan consists of a wooden tub, 15 ft. diameter and 30 in. deep, through the center of which projects

a small conical casting for the passage of the working shafts. Practically the entire bottom of the pan is covered by an amalgamated copper plate, to which quicksilver is automatically added during operation, in proportion to the amount of precious metal in the ore being treated. About 9 in. above the copper, or cathode, plate, a cast-iron anode plate equal in area to the copper plate is mechanically suspended. The cast-iron plate is divided into two annular rings, the outer one containing 60, and the inner one 40 per-cent. of the anode area.

These annular anodes are revolved in the same direction, and independently, by means of pulleys, gears, and a solid and a hollow shaft, so that any midway point of the inner ring travels practically the same number of feet per minute as any midway point of the outer ring. On the lower side of each of the anode rings, wooden paddles 7 in. deep are fastened radially, about 5 ft. apart, the bottom of the paddles being 2 in. above the copper plate.

The compound rotary motions of the anode rings and their paddles give to the pulp confined between them and the cathode plate a very uniform spiral motion outward, keeping the ore in a perfect state of suspension, and at a little higher speed than pulp travels down a stamp-battery plate. By the centrifugal force of the paddles, the pulp on reaching the staves of the pan is forced up between them and the outer edge of the outer anode to a height of about 16 in. above the anode, retaining the rotary motion imparted by the paddles and now endeavoring, by gravity, to seek the lower level at the center of the pan; the resultant motion is a downward and inward spiral until the ore passes over the inner edge of the inner anode, near the center cone, when it again meets the paddles and starts to repeat its cycle of travel.

At the periphery of the pan, and on a level with the cathode plate, 20 jets of compressed air are admitted through needle valves set at regular intervals around the pan. The air pressure — about 12½ lb. per sq. in. — is sufficient to overcome the centrifugal force of the pulp against the valve exits and to keep them clear. The amount of air is calculated from the oxygen demanded by Elsner's formula for dissolving the precious metals by cyanide salt, with an allowance for unavoidable waste in mixing. The air, on leaving the valve, travels against the outward spiral motion of the pulp

between the anode and cathode plates, and nearly all the excess makes its appearance as a slight effervescence around the center cone, being thoroughly incorporated with the pulp; comparatively little escapes by rising to the surface without passing between the plates. At 100 lb. pressure, 1½ ft. of air per min. is an ample supply for the pan during treatment.

The writer, during the past eight years, has experimented with many iodine, bromine and other chemicals, seeking economy and efficiency, and has found nothing so effective and economical as commercial potassium or sodium cyanide, or the commercial double salt, together with common lime and common salt, aided by the sodium amalgam and caustic soda generated by the process.

The electrical factor of the process, which consumes from 75 to 80 percent. of all the physical power necessary to operate it, and accounts for the speed, efficiency and economy of this as compared with other processes, is worthy of considerable attention.

From a 10 to 12-volt, 400-ampere, D. C. generator, the positive wire, after leaving the switch and indicator board, is led to the mechanical portion of the pan in four branches, each of which terminates in a suitable brush; two play against the copper collecting ring on the solid shaft which drives the outer annulus of the anode, and two against a similar ring on the hollow shaft which drives the inner annulus of the anode. By this means, the electric current passes up the vertical driving shafts, insulated from the cone through which they pass, and proceeds down the driving arms and supporting rods to the anode annuli, where it is uniformly distributed.

The negative wire from the electric generator, after passing through the switch and indicator board, also terminates in four branches, each of which connects with a copper tail piece projecting through the pan, each tail being a portion of one of the four segments of the copper plate forming the cathode. Owing to the insulation of the machine, all the electric current must pass from anode to cathode through the pulp.

In treating a charge consisting of 7½ tons of ore (dry weight) with an equal weight of water, and containing its quota of lime previously added during the crushing of the ore, such an amount of strong cyanide solution is added as to equal 1 lb. of cyanide salt

per ton of ore in the pan. The correct weight of cyanide per ton of ore may be previously determined by titrating filtered samples from the pan charge during a test run for cyanide consumption. After addition of the cyanide, finely crushed common salt is added until the ammeter registers 300 to 500 amperes, at 9.5 to 10 volts, as determined by a test run. The pan is then put in motion, the air is turned on, and the quicksilver is put into its distributor.

Many complex chemical compounds are formed, some of which are decomposed, in a manner which baffles the skill of the electro-chemist to explain, but among the established effects of the electric current the following may be stated:

The salt is decomposed by the electric current and metallic sodium is constantly precipitated on the mercury-coated copper plate, forming sodium amalgam; the sodium is converted into caustic soda, by its reaction with water, which, uniting with acid radicals in the ore, protects the cyanide from compounds from which the lime does not so readily protect it, thus saving a much larger portion of cyanide salt than can be saved without the current. Chlorine is also generated, but rarely indicates its presence, probably being united with basic radicals of the complex charge.

These several ions travel very slowly in quiet solutions, but in this mechanical device the pulp travels an average of seven miles over the copper plate in a 2½-hour treatment, presenting the cathions rapidly for discharge and deposition. This is analagous to the method by which copper is precipitated on a platinum dish, in modern electro-analysis for copper, by a revolving anode in a small fraction of the time required for precipitation in quiet solutions.

The cyanide anion probably seizes a potash or soda radical and is reconverted into nascent cyanide salt, unless a particle of precious metal, for which it has a greater affinity, presents itself first. However, there can be no question as to the so-called restoration of cyanide, for the writer has frequently found much more free cyanide of potash or soda in a sample of solution taken a half hour or an hour after a previous sample.

Theoretically no mercury can be lost by scouring or flouring in the charge, as any subdivided particle of this metal too small to seek the cathode by gravity is sufficiently small to yield to electro-

cyanide solution, whereby it is electro-chemically precipitated back on the plate. Practically, a small amount is lost, for in a 2500-ton run on Ruth ore a loss of about 5 lb. was reported.

On samples containing 0.8575 oz. of gold per ton, the writer has reduced the average tailing sample to 0.0075 oz. per ton, showing an extraction of 99.1 percent. by six hours' treatment in the electric pan, using 1 lb. KCN per ton of ore; the same ore, reduced to the same fineness, yielded but 92 percent. to solution in 48 hours' treatment in Pachuca tank (generally recognized as one of the best aero-agitating devices) using three tons of solution containing 3 lb. KCN per ton, or 9 lb. of cyanide salt per ton of ore.

Many of our western precious-metal ores from oxidized zones are contaminated by carbonates of lead, zinc and copper, which preclude economic treatment by the ordinary cyanide methods, owing to the difficulty in precipitating the precious metals in the presence of the cyanides of the base metals, aside from the excessive consumption of cyanide salt. The electric current, however, precipitates the base metals as well as the precious metals, on the amalgamated plate. The writer has produced, by the electric pan, bars of bullion worth $1200 or $1400 which averaged but 512 fine in precious metals, the other 50 percent. of the bars being lead, zinc and copper, alloyed with the precious metals. The refining charges on such bullion are low.

As precipitation of precious metals takes place more rapidly from strong than from weak solution, in treating certain ores economy can be introduced by discharging the pan at a fixed value of unprecipitated precious metals in solution (50 c. to $1 per ton). After passing the charge through a revolving settler used as a precautionary amalgam trap, it may then be sent through a Dorr thickener, whereby 50 percent. of the solution from a 1:1 charge, and 80 percent. of that from a 1½:1 charge, can be returned clear for original charges; this not only saves time and increases the daily capacity of the plant, but secures a higher percentage of recovery of the precious metals, while preventing undue waste of common salt and unused cyanide.

The total horse-power required for a single pan is 0.6 h.p. for mechanical agitation; 1 to 1.5 h.p. for compressed air; and 4.5 to 6 h.p. for electric current.

The capacity of the pan depends on the amount and character of gangue in the ore and on the nature of its precious metal minerals. The proportion of gangue determines whether one ton of ore may be treated with 1 ton or will require 1½ tons of water; none but the very clayey types require the 1 :1½ charge.

A single pan charge of the 1 :1 class is 7.5 tons of ore, dry weight, and of 1 :1½ ore, is 6 tons; the time required for treatment varies from 1 to 12 hours, generally being 2½ to 4 hours; hence the capacity of the pan, allowing for charging and discharging, varies between limits of 15 and 90 tons per diem, but on the average run of ores it varies from 36 to 60 tons per diem.

The large capacity of the pan, the small tonnage of ore under treatment in the mill at one time, and the small amount of machinery and apparatus required, reduce the total cost of installing mills using this process to 30 to 60 percent. of the amount necessary to install any other process of equal daily capacity.

A few examples will show the efficiency of the process.

Ruth Gold Mining Company's mill, Kingman, Arizona.—Daily capacity 50 tons. Quartz ore with large amount of manganese minerals; little free gold. Values occur in sulphide minerals; ratio of gold to silver 1 to 12. Test in 40-in. testing plant. Pan charge, 215 lb.; 1 lb. KCN per ton of ore; six hours' treatment; 200-mesh pulp. Results: Precipitated as amalgam, gold 96.76 percent.; silver 87.7 percent.; monetary values 95.6 percent. Shortly after the mill was turned over to the Ruth company, three pans being used, each with 7½-ton charges, the mill foreman's samples and his assayer's results showed as follows: Precipitated as amalgam, gold 96.66 percent.; silver 87.77 percent.; monetary values 95.09 percent.; 1 lb. KCN per ton of ore, six hours' treatment; value of ore $18 per ton.

Maginnis mill, Maiden, Montana.—One pan; 7½-ton charge; 30-mesh pulp. Oxidized ores containing 0.33 to 0.50 oz. gold and from 1.5 to 4 oz. of silver per ton, contaminated with carbonates of lead, zinc and copper. One pound KCN per ton; 2½ hours' treatment. Precipitated as amalgam, 86 to 92 percent. gold, 62 to 72 percent. silver. The low silver contents of this ore prohibited the economic use of cyanide salt in stronger solutions, in order to secure higher extraction from its silver minerals.

Gold ores from Kendal, Fergus County, Montana.— Decomposed, porous limestone, impregnated with gold; an ideal cyanide ore. Test of 225 lb. in testing pan; 12-mesh; 1 lb. KCN per ton; after one hour, 90.5 percent. of gold as amalgam; after 2½ hours, 97 percent. of gold contents of the ore reduced to amalgam. Ore valued at $8 per ton in gold.

Orient, Washington.——Auriferous, fine crystals of pyrite in andesite; no free gold. Average value, 1.03 oz., $21.29 per ton. Four hours' treatment on 230-lb. lots; 4½ lb. NaCN; pulp 120-mesh. 95.35 percent. of gold contents saved as amalgam.

Whitman mine, Pearl, Idaho.—Ore contained arsenical pyrites, 25 to 27 percent.; pyrite 40 percent.; gangue 35 to 37 percent. Six hours' treatment with 4 lb. KCN; 120-mesh. Gold, 1.3385 oz. per ton, worth $27.67; saved in form of amalgam, 88.8 percent.

Metates mine, Sinaloa, Mexico.— Ore contained 5 percent. pyrrhotite, with galena and antimonial sulphide minerals, in quartz gangue. Twelve hours' treatment; 230 lb., 120-mesh; 12 lb. KCN used; 7.7 lb. consumed. Gold, 0.625 oz.; silver, 63.015 oz.; value, $44.43 per ton. Extracted in form of amalgam, 99 percent. gold, 91.1 percent. silver.

All of the results cited above were obtained by direct pan treatment, without returning any solution once passed through the pan.

NOTES ON RARE MINERALS FROM CALIFORNIA.

BY AUSTIN F. ROGERS.*

1. Native Lead from Butte County.

I am indebted to E. W. Rice for some native lead found at a prospect 14 miles east of Chico on the West Fork of the Feather River. The lead occurs as small (3 or 4 mm.) sub-angular fragments covered with a clay-like substance. The mineral is sectile and malleable, and gives tests for lead. Nothing more is known as to its mode of occurrence but it certainly is not artificial, on account of the shape of the fragments.

2. Metacinnabarite from San Benito County.

A specimen from the Picachos mine consists of black, massive metacinnabarite surrounded by cinnabar, occurring on siliceous gangue. The metacinnabarite is completely volatile when heated on charcoal, and in the closed tube with soda yields globules of mercury. This specimen was presented to Stanford University by W. E. Stuart, of San Francisco.

Metacinnabarite has also been reported from New Idria, from New Almaden, from the Baker mine near Knoxville, and from the Reddington mine in Lake County.

3. Sal-ammoniac from Santa Barbara County.

A. D. Shepard sent specimens from burning oil shales on the Hope ranch, Santa Barbara County. The sal-ammoniac occurs in crusts 5 mm. thick, with columnar structure, and is associated with sulfur. It is recognized by its sharp, biting taste. It recrystallizes from water solution in branching crystal aggregates. The rock on which these minerals occur is evidently the Monterey shale.

4. Sal-ammoniac from Los Angeles County.

F. E. Vaughan gave me a specimen of sal-ammoniac from Burn-

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ing Mountain, a locality northwest of Santa Monica. The sal-ammoniac occurs as an incrustation on Monterey shale and is a sublimate produced by the spontaneous combustion of organic matter in the shale. The crystals are colorless and 2 or 3 mm. in size. As usual with sal-ammoniac the crystals are much distorted, the distortion being in the direction of one of the axes of three-fold symmetry, thus giving a triangular or tetrahedral appearance. The cube is a prominent form.

5. Tridymite from Tuolomne County.

The writer found this mineral in cavities of an andesite near Jamestown. The andesite occurs in large water-worn boulders in the bottom of the ancient river channel beneath the Table Mt. latite flow, the boulders having been exposed by drift mining. The tridymite consists of very thin, translucent white, hexagonal plates of about ½mm. maximum size. The double refraction is weak and the extinction is in confused sectors. There is a narrow border around the crystal with definite extinction parallel to the outline.

Tridymite has been reported from Mono County by Schaller.[1]

6. Valentinite from San Benito County.

This mineral from the Picahotes mine occurs in lemon-yellow, bladed aggregates with a cleavage parallel to the length of the crystal. The luster is pearly. The valentinite is probably pseudomorphous after stibnite. The associated minerals are cinnabar, quartz, and chalcedony. The mineral was identified by optical and by blowpipe tests. These specimens were furnished by W. E. Stuart.

7. Aurichalcite from Inyo County.

At the Cerro Gordo mine, aurichalcite occurs in pale, bluish-green, acicular crystals and tufts, as a secondary mineral in cavities, associated with calamine.

8. Hydrozincite from Inyo County.

This mineral also occurs at the Cerro Gordo mine. Among specimens presented by G. E. Gamble was a white, chalky mass of hydrozincite, which gave chemical tests for Zn, CO_3, H_2O and very faint

[1] Bull. 262, U. S. Geol. Surv., p. 129, 1905.

tests for Fe and Al. It contains a little calamine in radial aggregates.

9. MICROLITE AND PRYOCHLORE FROM SAN DIEGO COUNTY.

R. M. Wilke gave me for identification a honey-yellow mineral occurring with albite, lepidolite, tourmaline, and colorless apatite. A few octahedral crystals with narrow faces of $\{011\}$ and $\{311\}$ were noted. The mineral has resinous luster and irregular fracture. It is infusible and unaltered in the closed tube. Crushed fragments are dark between crossed nicols.

With the microlite is associated what is believed to be pyrochlore though no distinct crystals were found. It is dark brown and is surrounded by the microlite. Fragments are translucent brown, and isotropic. It is infusible, but turns yellow and glows, and gives a uranium test in a $NaPO_3$ bead.

10. LAZULITE FROM MONO COUNTY.

T. J. Hoover found lazulite in a quartz vein one mile west of Green Lake, in Green Creek Cañon near Bodie. The lazulite is found in large deep blue anhedra. The only associated minerals are quartz and muscovite.

11. SCORODITE AND PITTICITE FROM MARIPOSA COUNTY.

The specimen containing these rare minerals was collected by H. W. Turner on the west side of the South Merced River, just below the mouth of Devil's Gulch. The specimen consists principally of massive arsenopyrite, the scorodite and pitticite being secondary minerals. The scorodite occurs as pale green crystals in cavities. The pitticite is a dark brown, amorphous mineral which resembles limonite but gives tests for the (AsO_4) and the (SO_4) radicals.

12. LUDWIGITE FROM EL DORADO COUNTY.

Among other minerals collected by the writer at the Consumnes copper mine, near Fairplay, El Dorado County, was a black mineral substance which proved to be ludwigite. It occurs in scaly, rather than fibrous masses, associated with calcite, epidote, molybdenite and chalcopyrite. Crushed fragments are green or brown plates, often of the greenish-brown color common in tourmaline sections, but apparently without pleochroism. The ludwigite is fusible to a

black, slightly magnetic glass. With boracic acid flux it gives a green flame indicating the presence of boron. The mineral is slowly soluble in hydrochloric and gives wet tests for iron and magnesium. The borax fusion gives tests for both ferrous and ferric iron.

13. BROCHANTITE FROM CALAVERAS COUNTY.

Brochantite occurs as druses of small, dark green crystals associated with, and derived from, chalcopyrite, at Copperopolis. It is soluble in nitric acid without effervescence and gives a microchemical test with calcite.

14. KALINITE FROM INYO COUNTY.

On the sides of a steaming vent two miles east of Coso Springs, C. A. Waring collected a white crust 2 cm. thick. This was tested and found to be kalinite, or native potassium alum.

15. JAROSITE FROM SAN BENITO COUNTY.

S. C. Herold collected this mineral on the 250-ft. level of the New Idria quicksilver mine. It occurs as a drusy coating of minute yellowish-brown crystals on a dark siliceous gangue, and was associated with cinnabar. The crystals are equidimensional, with the forms $\{1001\}$, $\{10\bar{1}1\}$, and $\{02\bar{2}1\}$ and simulate cubo-octahedrons. Some crystals are dark between crossed nicols but most of them are pleochroic. They give the chemical and pyrognostic tests of jarosite, and contain very little sodium.

16. ADULARIA FROM ELDORADO COUNTY.

A specimen from the south side of Fallen Leaf Lake, collected by Professor J. C. Branner, contains colorless adularia crystals of about 5 mm. size. The forms are $m\{110\}$, $x\{\bar{1}01\}$, $c\{001\}$, and $b\{010\}$. The habit is short prismatic with x as a prominent form. With the adularia is associated diopside in pale green prismatic crystals with the following forms: $a\{100\}$, $b\{010\}$, $m\{110\}$, $c\{001\}$, and $\{hkl\}$. So far as I know this is the only locality for adularia within California.

17. VALENCIANITE FROM CALAVERAS COUNTY.

Valencianite is the name applied to vein orthoclase. It has the

habit of adularia but is usually translucent or opaque, like ordinary orthoclase. Five miles east of Milton, on the road to Copperopolis, I found crystals of valencianite in seams of what appeared to be a decomposed igneous rock. The crystals are prismatic in habit with $m\{110\}$, $x\{10\bar{1}\}$, c, $\{001\}$ and $b\{010\}$. The habit is that of adularia but the appearance is that of ordinary orthoclase, being reddish opaque.

18. Crocidolite from Santa Clara County.

In the Franciscan rocks of the Coast Ranges, siliceous rocks containing a blue, fibrous, glaucophane-like mineral are often found, especially as float. An exceptionally good specimen of the mineral, found in the region east of Mt. Hamilton, but exact locality unknown, was analyzed by A. K. Schellinger, with the following results. SiO_2, 50.65; Al_2O_3, 0.90; Fe_2O_3, 19.21: FeO, 21.70; MgO, 0.79; CaO, 0.39; Na_2O, 4.93; H_2O, 1.43. Total, 100. This places the mineral as crocidolite. It occurs as fibrous seams about 4 mm. thick in a metamorphic rock.[2]

19. Lapis Lazuli from San Bernardino County.

R. M. Wilke found a small boulder of lapis lazuli in this county. The specimen is essentially a dark gray, fine-grained crystalline limestone. Crushed fragments examined under the microscope show calcite, lazurite in deep blue irregular grains, pyrite, diopside, and an unknown mineral with rather weak double refraction.

20. Californite from Butte County.

A. K. Schellinger kindly presented to Stanford University several pebbles of californite from the gravels at the mouth of the Feather River, at Oroville. These specimens are pale green, water-worn pebbles varying from 1 to 3 in. diameter. The specific gravity of one pebble was found to be 3.25. A thin section proves the composite nature of the californite; four or five minerals are present. An almost colorless mineral, without structure, and with moderate relief, is vesuvianite. With crossed nicols it shows interlocking areas with a peculiar, greenish-gray, anomalous interference

[2] Louderback has also recorded this mineral from the Coast Ranges. *Bull.* Geol. Soc. Am., Vol. 18, p. 659.

color. An isotropic mineral with the same relief as vesuvianite, and scarcely distinguishable from it, is garnet. There are also a few anhedra of diopside, a little calcite, and a mineral with aggregate structure and low relief which is probably chalcedony.

21. Piedmontite from San Diego County.

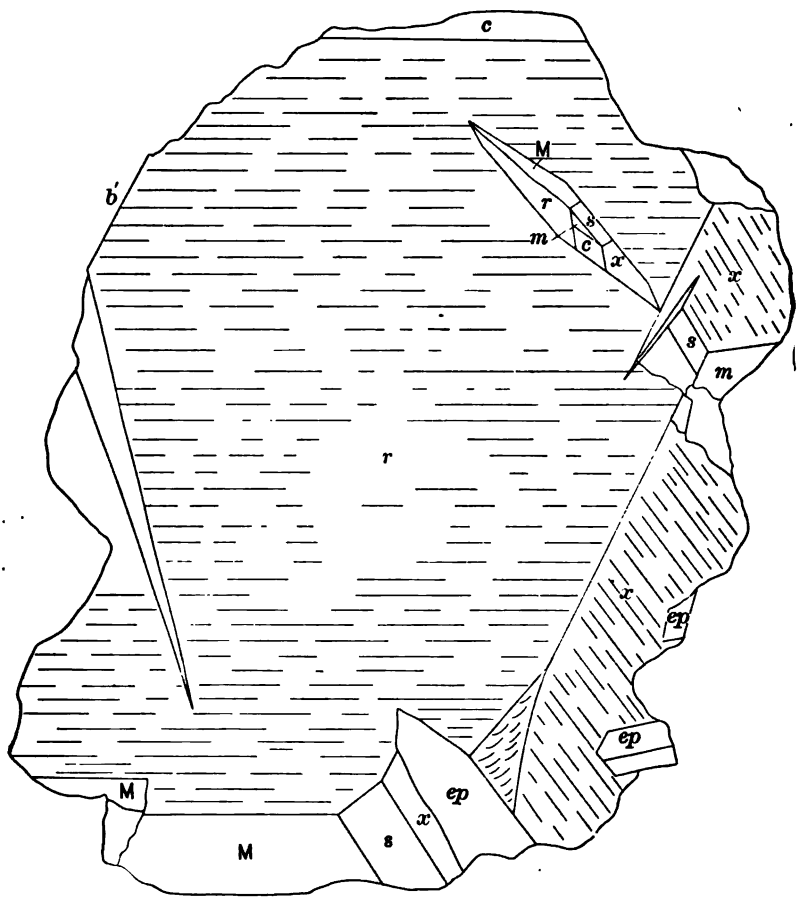
This rare mineral, a manganese epidote, has heretofore been found at but two localities in this country, South Mt., Penn., and Annapolis, Mo. I found the mineral in a thin section of a quartz-porphyry boulder from the Pliocene or Pleistocene gravels at Pacific Beach, San Diego County. The rock is a dark red porphyry containing quartz, orthoclase, and plagioclase, in a cryptocrystalline groundmass. The piedmontite is a secondary mineral replacing some original ferro-magnesium mineral, probably biotite. It is recognized by the purple red color, and the pleochroism (purplish red to orange yellow). With the piedmontite is associated epidote, also secondary, and black grains of manganese dioxide. The red color of the rock is due principally to finely divided hematite.

22. Axinite from Riverside County.

Hon. Edward Hyatt, State Superintendent of Public Instruction for California, kindly lent me, for study, the unique axinite crystal here described. Mr. Hyatt furnishes the following paragraph concerning its occurrence: "This specimen came from what is known as the city stone quarry, within the corporate limits of the city of Riverside. It is on what is called North Hill, near Fairmont Park. The quarry furnishes material for the city stone-crushing plant. The chief rock used is a white, crystalline limestone, which is associated with hornblendic syenite and granite, and with magnetite, and many other minerals. This specimen came from a large cavity, filled with dried clay, which was found about midway in the quarry."

The axinite is a crystal of tabular habit about 9 x 12 cm. and about 1½ cm. thick, and is unique on account of its size. The largest crystal mentioned in Hintze's *Handbuch der Mineralogie* measures 5 cm. It is rather irregular in outline but the faces are fairly well developed. The accompanying figure, which is natural size, is a fairly accurate drawing with the irregularities sketched in

The forms are $r\{1\bar{1}1\}$, $x\{111\}$, $M\{1\bar{1}0\}$, $s\{201\}$, $c\{001\}$, $m\{110\}$, and $b\{01\bar{0}\}$. The forms were identified by the following measurements (each the average of three) made with a contact



AXINITE FROM RIVERSIDE, CAL.
(Actual Size)

goniometer: $xr = 39°\ 20'$ (calc.$= 40°\ 46'$); $rM = 45°\ 50'$ (calc.$= 45°\ 15'$); $rs = 36°\ 40'$ (calc.$= 36°\ 25'$); $mx = 30°\ 53'$ (calc.$= 30°\ 33'$); $Ms = 34°\ 10'$ (calc.$= 33°\ 18'$); $xM = 48°\ 03'$ (calc.$=$

49° 25′); $cM' = 90°\ 50'$ (calc.$= 90°\ 4\frac{1}{2}'$); $rb' = 91°\ 40'$ (calc. 93° 21′). The habit is tabular, parallel to $r\ \{\bar{1}11\}$; r, the dominant face, is faintly striated parallel to its intersection edge with c and M. The forms $x\ \{111\}$ and $M\ \{1\bar{1}0\}$ are fairly large but the other forms are very small and subordinate.

The sketch, near the upper part of the drawing, represents a projecting axinite crystal of thick tabular habit parallel to r but the subordinate forms c, s and m are better developed than on the large crystal. In the drawing, the letters *ep* represent enclosed epidote crystals. The only other associated mineral is byssolite, which is found in some of the cavities.

The color of the axinite is violet-brown. It is translucent but subtransparent in spots. Polarized light, incident on r, and vibrating parallel to the *Mrc* edge, is practically all absorbed, as tested with a Nicol prism.

23. Kämmererite from Alameda County.

Kämmererite is a chromium-bearing chlorite. I have found this mineral associated with chromite at the Mendenhall chrome mines (now abandoned) on Cedar Mountain, about 16 miles southeast of Livermore. It is a reddish violet, micaceous mineral which gives in convergent light a biaxial interference figure with small axial angle. The largest specimen was a cleavage plate about 2 cm. in diameter.

24. Deweylite from Santa Clara County.

At the big magnesite mine on Red Mountain, in the extreme northeastern corner of Santa Clara County, one of the few minerals found was deweylite. It occurs as a crust, often botryoidal, on serpentine and resembles gum arabic. In part it is pseudomorphous after hexagonal prismatic crystals, probably aragonite. It is pale brownish-yellow and has a hardness of about 3. In fragments, between crossed nicols, it has the banded and radiate structure of chalcedony and shows an imperfect spherulitic cross. Dana[3] gives deweylite as amorphous but Lacroix[4] says it has the optical structure of chalcedony, with positive elongation. The specimens I have tested have positive elongation in some bands and negative in others.

[3] System of Mineralogy, 6th Ed., p. 676.
[4] Mineralogie de la France, Vol. I, p. 438.

When heated in the platinum forceps a fragment turns black, then white, but is practically infusible. When a large quantity of the powder is heated in a test tube, a gas is given off which can be ignited. This indicates the presence of organic matter. The mineral is soluble with difficulty in hydrochloric acid and the solution gives a test for magnesium.

25. Halloysite from Inyo County.

A banded, white and pale brown, clay-like mineral from the Cerro Gordo mines proves to be halloysite. It adheres strongly to the tongue. A rough water determination gave 21 per cent., the theoretical being 19.6 per cent.

Halloysite has been recorded from San Diego County by Schaller.[5]

26. Agalmatolite (Pyrophyllite) from San Diego County.

A specimen from near Encinitas was given to the writer by R. M. Wilke for identification. It is a compact, massive substance, with body color of cream, with dark red streaks, the streaks often being interrupted by circular spots. It has a specific gravity of 2.77, is fusible on the edges, and gives a blue color when ignited with cobalt nitrate solution. It is practically insoluble in acids but the acid solution of the soda fusion gave a good test for aluminum and only faint tests for calcium and magnesium. The mineral is harder than talc but is easily cut by a knife and takes a good polish.

Stanford University, California, Dec., 1911.

[5] *Bull. 262*, U. S. Geol. Surv., p. 121, 1905.

LIMONITE DEPOSITS OF STATEN ISLAND, NEW YORK.

BY CHARLES R. FETTKE.

Although the limonite deposits of Staten Island have been practically exhausted and will probably never be worked again, considerable scientific interest has attached to them recently on account of their similarity to the brown iron ores of Cuba, which have attracted a great deal of attention within the last three years. The Cuban deposits cover a vast area and promise to become large contributors to the world's production of iron ore in the near future.

Topography of Staten Island.

Staten Island comprises the most southern portion of the State of New York. It is about 13½ miles long and 7 miles wide at the widest part. The most marked topographic feature is a ridge which extends from the town of Richmond, at about the center of the island, in a northeasterly direction to New Brighton at the northeast extremity. This ridge reaches a maximum elevation of about 375 ft. On the southeast it is bounded by a steep escarpment which separates it from the adjoining coastal plain occurring on this side of the island, while on the northwest side it slopes off irregularly but gradually to the level of the salt water marshes bordering on Kill van Kull and Staten Island Sound, which separate the island from the mainland.

Summary of the Geology.

The geology of Staten Island is quite simple. The high ridge is made up of serpentine. The northwestern corner of the island is underlain by sandstones and shales, with an intercalated intrusive sheet of diabase. These sandstones and shales belong to the Newark formation and are believed to be of late Triassic and early Jurassic age. The contact between the serpentine and the Newark formation is not exposed. From evidence gathered elsewhere, however, it is known that the serpentine is much older. The entire

area south of the serpentine ridge is probably underlain by sands and clays of Cretaceous and later systems.

The greater portion of Staten Island is covered by glacial drift. The terminal moraine of the Pleistocene ice sheet extends in an irregular line from the Narrows at the northeastern end of the island to Tottenville at the southwestern extremity. At one place in the vicinity of Garretsons' the terminal moraine bends to the northward and covers a portion of the serpentine ridge. Here is a small area not covered by drift. North of this terminal moraine the greater part of the island is covered by the ground moraine. In many places it is quite thin, while in others it varies up to as high as 84 ft. thick.

The limonite deposits are directly associated with the serpentine. This rock has been the subject of a careful study by D. H. Newland.[1] He came to the conclusion that it was derived from a basic, igneous, intrusive mass allied to the pyroxenites and peridotites. It is older than the Triassic sandstone and of more recent origin than the Manhattan schist, whose exact age is still uncertain, but belongs either to the pre-Cambrian or to the early Palaeozoic. This basic igneous rock was probably intruded into the Manhattan schist in the form of a boss, and underwent serpentinization while still under comparatively deep-seated conditions. Through uplift and erosion it was finally brought to the surface where it has probably been exposed to the action of weathering ever since the Cretaceous period. Since that time, several periods of uplift and subsidence have occurred, but there is no evidence that the area occupied by the serpentine ridge was ever submerged below sea-level again.

Distribution and Occurrence of the Limonite Deposits.

The limonite deposits rest on the serpentine and are in turn covered by the glacial drift. Those in the vicinity of Grant City and Garretson's lie south of the terminal moraine, and, therefore, the covering of drift is absent in places. Deposits of economic value were found in the vicinity of Grant City, at Castleton Corners, and near the top of the ridge along Ocean Terrace Road.

[1] "The Serpentines of Manhattan Island and Vicinity, and their Accompanying Minerals," D. H. Newland. School of Mines Quarterly, Vol. 22.

Grant City is situated at the foot of the southeastern escarpment of the serpentine ridge while Castleton Corners lies well down toward the foot of the northwestern flank. This shows that the deposits are scattered over a wide area within the serpentine belt.

The old New Dorp mine is situated on the hill north of Grant City. B. T. Putnam, in his report for the Tenth Census of the United States,[2] describes this deposit as consisting of 4 to 5 ft. of limonite resting on serpentine, from which it was separated by a few inches of ocher. At the west end it was covered with only an inch or two of soil, but toward the east the surface material thickened. The pit has an approximate area of 1½ acres. The hill slopes off quite abruptly to the south and west of it, while to the northeast the surface is a gradually rising one. About 250 ft. northeast of the New Dorp mine is another opening known as the old Towle mine. It covers an area of less than 1-10 acre. Here the limonite overlying the serpentine had a thickness of 8 ft. and was overlain by 10 ft. of gravel, sand, and clay. Both of the above mines are a short distance south of the terminal moraine.

About 1½ mile north of the above mine, at the summit of the serpentine ridge along Ocean Terrace Road, is another opening known as the Cooper and Hewitt mine. Here 5 ft. of limonite were overlain by 7 ft. of clay and gravel. The pit has an area of about ⅓ acre. A short distance east of this opening, on the east side of the road, is another opening which has an area of about one acre. No data are available as to the thickness of the limonite encountered here. It is situated on a small ridge which has fairly steep slopes on three sides.

The old Tyson mine at Castleton Corners constitutes another important opening. It has an approximate area of 3¼ acres. Here the bed of limonite had a thickness of 6 to 8 ft. and was covered by 9 ft. of clay and gravel. Other openings of minor importance have been made at other points on the ridge but the above are the larger ones from which ore has been taken.

The relationship of the limonite to the serpentine is shown in a gully north of Grant City. At the lower end of the gully the comparatively fresh serpentine is exposed. It is here overlain by

[2] Tenth Census of the United States, 1880, Vol. XV. "Mining Industries of the United States excluding Precious Metals." B. T. Putnam.

a soft chloritic and talcose mass. Further up the gully weathered serpentine is exposed. It has been considerably softened and rendered porous. The dark green color has been replaced by a light yellowish green. Considerable talc is present here in the serpentine. The altered serpentine becomes more and more stained with the red oxide of iron toward the surface, until it finally grades into a dark red, earthy mass. Crusts and pisolitic masses of limonite appear at or near the surface. These crusts of limonite often include small flakes of talc.

The Iron Ore.

The limonite of Staten Island occurs in the form of cellular and spongy masses, botryoidal crusts, and concretionary grains, associated with a yellowish brown and red earthy mass. The botryoidal crusts of limonite often have a smooth, black surface and frequently show a fibrous radiated structure on fracture. This variety is probably turgite, rather than true limonite. The concretionary grains of limonite consist of little pellets varying in size from $\frac{1}{8}$ to $\frac{1}{4}$ in. and even more in diameter. These are imbedded in a ferruginous matrix, giving the whole mass a pisolitic texture like that so frequently developed in bauxite deposits.

B. T. Putnam[2] quotes the following analysis of a sample of limonite taken from a pile of 3,000 tons of washed ore at the New Dorp mine:

SiO_2	14.19 %
Al_2O_3	8.59
Fe_2O_3	55.95
FeO	0.55
MnO	0.52
MgO	3.60
CaO	0.18
K_2O	0.19
Na_2O	0.40
Cr_2O_3	1.91
FeS_2	0.293
NiS.	0.55
CoS.	0.17
CO_2	0.22
P_2O_5	0.135
C (in carbonaceous matter).	0.13
H_2O (Hygroscopic)	2.21
H_2O (Composition)	10.20
Total	99.988

Sulphur	0.39 %
Phosphorus	0.059
Metallic Iron	39.72

Insoluble Siliceous Matter.

SiO_2	14.19%
Al_2O_3	3.47
MgO	2.92
K_2O	0.03
Na_2O	0.07
P_2O_5	0.01
Cr_2O_3	0.68
Total	21.37

[2] Op. cit.

If this analysis is recast into the percentages by weight of the different minerals present, the following results are obtained:

Limonite, $2Fe_2O_3.3H_2O$	64.52%
Gibbsite, $Al_2O_3.3H_2O$	7.80
Quartz, SiO_2	4.25
Magnesite, $MgCO_3$	1.85
Kaolinite, $Al_2O_3.2SiO_2.2H_2O$	8.77
Talc, $H_2O.3MgO.4SiO_2$	9.07
Apatite, $9CaO.3P_2O_5.CaCl_2$	0.34
Chromite, $FeCr_2O_4$	2.70
Pyrite, FeS_2	0.29
Millerite, NiS	0.55
Cobalt Sulfide, CoS	0.17
Carbon, C	0.13
	100.44

The small amounts of sodium and potassium were neglected in this computation. The insoluble alumina was calculated as kaolimite and the soluble as gibbsite. The insoluble magnesia is present mostly as talc. Small flakes of this mineral can often be seen scattered through the ore. The percentage of water as shown by the analysis, including the hygroscopic water, lacked 1 percent. of being sufficient to give the amounts of limonite and gibbsite indicated by the above calculation. It is very likely, therefore, that less hydrated forms of ferric oxide and alumina were also present.

The conspicuous features of this ore are the rather high percentage of alumina, a considerable part of which evidently occurs as the hydrated oxide, and the comparatively large amount of chromium. Part of the silica is present as free quartz. The percentage of phosphorus is quite low. It is also interesting to note that the presence of small percentages of nickel and cobalt was detected in the ore.

Associated with the limonite deposits, large masses of quartz, often reaching a diameter of 4 ft. or more are found at times. These have a cellular and a spongy structure. When examined closely it is seen that they are composed almost entirely of crystalline and chalcedonic quartz.

Genesis of the Deposits.

Several explanations as to the origin of the limonite deposits of

Staten Island have been given by various observers. W. W. Mather[3] in his report on the geology of the first geological district of New York called attention to the fact that they overlie a serpentine rock which contains an abundance of carbonate and hydrate of magnesia. This, he thought, performed the same office in the deposition of the ore as did the limestone beds of Columbia and Dutchess counties in the limonite deposits occurring there. N. L. Britton[4] thought that the limonite was deposited by thermal springs which came to the surface through crevices in the serpentine, the iron being in solution as carbonate, which became oxidized as soon as it came in contact with the atmosphere. T. Sterry Hunt,[5] came to the conclusion that they were residual deposits resulting from the weathering of the serpentine rock during a period antedating the glacial erosion. Another view held by some observers[6] is that the limonite was deposited in superficial hollows of the serpentine by ferruginous waters which had leached the iron out of the adjacent rock and come to rest in these depressions.

The fact that the deposits nearly always occur on the flanks or tops of ridges favors a residual method of origin rather than precipitation from solution in depressions. If they had been formed in the latter way a great amount of erosion and an almost complete change of topography would have been necessary to bring them into their present position. The comparatively large percentage of chromite in the ore indicates a residuary deposit. Chromite is a refractory mineral and does not go into solution readily; therefore limonites deposited from solution ought to be comparatively free from it. The facts that the alumina and iron appear to be present in about the same ratio in the iron ore as in the serpentine, and that the unaltered serpentine can be seen in places to grade through various stages of weathering into the ovrlying iron ore also in-

[3] "Geology of New York," Part I, 1843, p. 489. W. W. Mather.
[4] "The Geology of Richmond County, New York." N. L. Britton. New York Academy of Science *Annals*, 1880-83, Volume 2, p. 161-182.
School of Mines Quarterly, Vol. 2, May, 1881.
[5] "Mineral Physiology and Physiography," 2nd edition, 1891, p. 268.
[6] "Geology of the City of New York." L. P. Gratacap.
"Ore Deposits of the United States and Canada," J. F. Kemp, p. 91.

dicate a residuary origin. A comparison of an analysis of the serpentine with that of the ore emphasizes this relationship:

ANALYSES OF SERPENTINE AND IRON ORE.

	Serpentine (D. H. Newland)[1]	Iron Ore (B. T. Putnam)[2]
SiO_2	36.72%	14.19%
Al_2O_3	1.06	8.59
Fe_2O_3	6.59	55.95
FeO	1.53	0.55
MnO	not reported	0.52
MgO	29.09	3.60
CaO	9.95	0.18
K_2O	not reported	0.19
Na_2O	" "	0.40
Cr_2O_3	0.49	1.91
FeS_2	not reported	0.293
NiS	" "	0.55
CoS	" "	0.17
CO_2	———	0.22
P_2O_5	not reported	0.135
C (In carbonaceous material)	———	0.13
H_2O (Hygroscopic)	0.52	2.21
H_2O (Composition)	14.02	10.20

A comparison of the two analyses shows that the iron ore contains 8 times as much alumina as the original serpentine, about 6½ times as much iron, and 4 times as much chromic oxide. Assuming that no alumina was lost, and that the serpentine had a specific gravity of 2.7 and the iron ore 3.7, it would take a thickness of 88 ft. of serpentine to yield 8 ft. of the iron ore. That means a removal of 95 percent. of the silica, 98½ percent. of the magnesia, and practically all of the lime.

It has been seen that the formation of these deposits dates back to pre-glacial times. As has already been stated, the serpentine was probably exposed to the action of weathering as far back as Cretaceous. During early Tertiary or Eocene time a humid, semi-tropical climate undoubtedly prevailed. During such favorable conditions this region must have been covered by a quite luxuriant growth of vegetation. These conditions are very favorable for the accumulation of such residual deposits as those described.

[1] Op. cit.
[2] Op. cit.

Warmth facilitates chemical decomposition. An abundance of plants, likewise, favors decomposition, but hinders mechanical disintegration, so that quite thick layers of decomposed material can accumulate. The absence of frost also greatly lessens mechanical disintegration. The process of weathering, therefore, was limited to chemical decomposition and solution.

The principal agents that brought about this decomposition of the serpentine were undoubtedly water, oxygen, carbon dioxide, and various organic acids derived from the decomposition of vegetable matter. These agents attacked the silicates of the serpentine, consisting principally of the mineral serpentine, with minor amounts of chlorite and talc, and broke them up into simple compounds. The calcium became carbonate and went into solution quite readily in the presence of carbon dioxide. The magnesium became carbonate or hydroxide, or both, and in this form could also be carried off in solution. The silica, being liberated in a colloidal and finely divided form, was quite readily soluble in the groundwaters containing organic acids and carbon dioxide. In the presence of the carbon dioxide and organic acids, there was probably a tendency for the iron to go into solution as ferrous carbonate. However, it would be oxidized and precipitated as ferric oxide as soon as it came into contact with waters containing oxygen, which must have been plentiful. The calcium carbonate in the presence of oxygen also, undoubtedly, had an important influence in precipitating the iron as hydrated ferric oxide. For this reason the iron did not migrate very far. Analyses seem to show that the greater part of the aluminum is also present as hydrated oxide. Colloidal alumina is soluble in weak organic acid solutions but was undoubtedly very readily precipitated from these unstable solutions and, therefore, was not transported far.

A residual mantle made up largely of hydrated ferric and aluminum oxides therefore resulted. Such deposits are known as laterites, and are very characteristic of humid, tropical and semi-tropical regions. In laterization, weathering is carried further than is usually the case in temperate zones. The silicates are more completely decomposed. leaching is more thorough, and the resulting residual product is richer in the hydrated oxides.

The conditions were probably favorable for the formation of these residual deposits on Staten Island throughout the greater part of Tertiary times and especially during the earlier Tertiary or Eocene periods. The deposits must have accumulated over the whole serpentine area that was exposed to weathering at that time. Since their formation they have suffered greatly from erosion, which probably removed large amounts of the residual mantle even during later Tertiary times when an uplift of the region of which Staten Island forms a part appears to have taken place and accelerated erosion. The greatest removal, however, was undoubtedly due to the advance of the Pleistocene continental glaciers over practically the whole of this area. The deposits that endured to the present era were only small remnants of a residual mantle which at one time covered the entire serpentine area.

Comparison with the Cuban Deposits.

Arthur C. Spencer[7] in his report on the brown iron ore deposits of Cuba was among the first to call attention to the similarity between those deposits and the deposits on Staten Island. The ore occurs as a nearly horizontal mantle overlying the serpentine country rock of the plateau region of central and northeastern Cuba. It varies in thickness from a few inches to 80 ft. with an average depth of 17 to 18 ft. Much of the ore is an earthy mass varying in color from dark red to yellow, with the lighter colored portions generally occurring near the bed-rock and the darker red at the surface. Near the top much of the ore also occurs as small shot-like particles which are often cemented together into boulders and masses having a pisolitic texture. All who have examined these deposits agree that they are residual in origin, being derived from the underlying serpentine by the weathering of this rock. A comparison of analyses of ores from Staten Island and from the Moa district of Cuba emphasizes the fact that they were undoubtedly derived by similar processes from the serpentine upon which they rest.

[7] "Three Deposits of Iron Ore in Cuba," *Bulletin* 340, U. S. G. S., p. 318.

	Staten Island Ore B. T. Putnam [2]	Cuban Ore J. S. Cox.[8]
SiO_2	14.19	6.26
Fe	39.72	45.67
Al_2O_3	8.59	10.64
Cr	1.31	1.96
Ni	0.36	0.845
Co.	0.11	
P.	0.059	0.008
S.	0.39	0.107
H_2O (Hygroscopic)	2.21	
H_2O (combined)	10.20	11.59

It has been estimated[9] that 1,900,000,000 tons of iron ore are available in the Cuban deposits and it appears that this estimate will probably be raised as development work goes on.

[2] Op. cit.

[8] "The Iron-Ore Deposits of the Moa District, Oriente Province, Island of Cuba," *Bulletin*, A. I. M. E., No. 51, March, 1911.

[9] "Iron Ore Resources of the World — West Indies." J. F. Kemp, pp. 794-795.

ABSTRACTS—ANALYTICAL CHEMISTRY.

BY E. WALLER.

APPARATUS — REAGENTS — MANIPULATION.

Electro Analysis — Stirring the Solution.—Fischer, Thiele and Stecher. (*Zts. Electrochem.*, XVII, 905) Instead of using a mechanical stirrer, it is proposed to bubble hydrogen gas through the electrolyte.

Standardizing Alkalimetric Solutions.— Klinkerfues. (*Chem. Ztg.*, XXXV, 1274) If pure Na_2CO_3 or other alkali is not available, one can dissolve a weighed amount of pure $(NH_4)_2SO_4$ and distil off all the NH_3 into a known volume of the acid to be standardized. The amount of acid used for this should be a little less than will suffice to neutralize all the NH_3, the test to be finished by adding the remainder of the acid from a burette.

Trimethylphenylammonium Carbonate.— Schirm. (*Chem. Ztg.*, XXXV, 1177 and 1193) This substance, $[C_6H_5(CH_3)_3N]CO_3$, is proposed as a substitute for Na_2CO_3 for precipitating Zn, Mn, Ni, Cu, etc. Although some of the base is probably occluded by the precipitate, it is volatile on ignition in contradistinction to the Na salt.

Hyperol.— Milbauer. (*Chem. Ztg.*, XXXV, 871) This is a new reagent in solid form (crystals or tablets) consisting of an H_2O_2 compound with urea, to which a little citric acid has been added to preserve it. Analysis showed it to contain 65.7 urea 34.0 H_2O_2 and 0.08 citric. It is soluble in water and alcohol; ether decomposes it with evolution of H_2O_2. It begins to decompose at 60° without any admixture. Appears to be applicable in most of the cases where H_2O_2 is ordinarily used as a reagent.

Delicate Tests.— Carney. (*J. Am. Chem. Soc.*, XXXIV, 32) The organic base, tetramethyldiaminodiphenylmethane $(CH_3)_4N_2(C_6H_4)_2CH_2$, which for brevity is called "tetra-methyl base," was used by Trillat as a test for Pb and Mn, the test being applied to the dry sulfates, after use of an oxidizing agent; a blue color appears on warming and disappears on cooling. Later, papers moistened with alcoholic solution of the base were used to distinguish O_3, NO and H_2O_2. O_3 gives a violet, NO a straw yellow, and H_2O_2 gives no coloration. Other methods of applying the test for Pb and Mn are described. The author finds that the base can also be used as a delicate reagent both for Au and for NH_3. The Au solution should be neutralized with Na_2CO_3, and then slightly acidified with citric, before adding the base. A blue to purple coloration is developed on warming. 0.01 mg. Au in 50 c.c. can be detected. To test for NH_3 the solution is boiled in a tube or flask with NaOH, and the escaping steam made to impinge on a bit of filter paper moistened with a weak solution of $MnSO_4$ and a drop of H_2O_2 solu-

tion. When much NH_3 is present a brown stain will show, without any use of the base. With but little NH_3 no discoloration is seen, but addition of a drop of solution of the tetra-methyl base develops a purplish or bluish spot. 0.02 mg. NH_3 or less can thus be detected.

Ferrous Salts — Qualitative.— Slawik. (*Chem. Ztg.*, XXXVI, 54) An alcoholic solution of dimethylglyoxime gives an intense red coloration in presence of tartaric, after the solution has been saturated with NH_3. Ferric salts do not give this reaction.

Ferrocyanide Determination.— Schroeder. (*Zts. anorg. chem.*, LXXII, 89) The method depends upon the destruction of the ferrocyanogen complex by hot $HgCl_2$ solution, the Fé assuming the ordinary ferrous condition, when it may be titrated. Dissolve 2 gm. of the sample in 50 to 100 c.c. cold H_2O. Add this to 100 c.c. cold saturated solution of $HgCl_2$. Add this mixture, a little at a time, as fast as the Hg precipitate disappears, to 200 c.c. of a boiling solution of $HgCl_2$ acidified with a few drops of HCl. Add the rinsings as well. Cool, add 30 c.c. Reinhardt's solution ($MnSO_4$ and H_3PO_4) and titrate with standard permanganate.

Peroxidized Compounds.— Leuchter. (*Chem. Ztg.*, XXXV, 1111) Description of the cobalt test: make first a 1% solution of $CoCl_2.6H_2O$; second, a solution of 1.6 parts borax, 20 parts glycerin, and 100 parts H_2O. For a test mix equal parts of these solutions in a slender test tube and pour gently on top of the mixture some of the solution to be tested, so as to form a layer. With peroxidized compounds a brown black layer forms at the point of contact of the liquids. In testing for persulfate the solution should be warmed.

Fluorides, Silicates and Fluo-Silicates—Qualitative.—Browning. (Am. *J. Sci.*, XXXII, Oct.) The use of moistened black filter paper to receive the action of the SiF_4 is recommended for delicacy. The material tested is heated in a lead cup with conc. H_2SO_4 in the usual way. The cup is covered with a sheet of lead having a hole in the centre on which is laid the moistened black paper. A whitish spot, due to formation of the hydrated SiO_2, is readily perceived. If SiO_2 has been added to the original material the test indicates HF or H_2SiF_6. If a fluoride has been added, it is a test for SiO_2.

Silicates — Decomposing by Hydrofluoric Acid.— Dittrich. (*Ber.*, XLIV, 990) In decomposing some silicates comparatively poor in SiO_2 and containing much MgO, as for example, olivine, by use of HF and H_2SO_4, the attack is facilitated by adding to the mineral an equal amount of "not too finely" pulverized quartz.

Ferric Oxide in Presence of Alumina.— Krieger. (*Chem. Ztg.*, XXXV, 1054) The precipitate of mixed oxides is treated, without pulverizing, in an Erlenmeyer flask with 25 c.c. conc. H_2SO_4, and a little water. Boil for 5 to 15 min., reduce with Zn, and titrate with standard permanganate. It is stated that the Al_2O_3 is scarcely attacked, while Fe is dissolved.

Ferrous Iron in Silicates.— Dittrich. (*Chem. Ztg.*, XXXV, 1093) After decomposing by the aid of HF, a titration with permanganate to determine FeO, existing as such, has frequently been found to give an

uncertain end reaction, which has been attributed to Mn or HF, or the presence of both together. The trouble is now attributed to Ti as Ti_2O_3. Addition of K_2SO_4 and pulverized SiO_2 to the solution is stated to remove the difficulty.

Nickel or Cobalt.— Dede. (*Chem. Ztg.*, XXXV, 1077) Instead of Br for separation as the higher oxides, persulfate in the cold is claimed to be quite as satisfactory. The solution should be only slightly alkaline after addition of the persulfate. Too much excess of alkali, or too dilute solutions, causes the action to be very slow, requiring several hours for complete precipitation.

Qualitative. Modified Scheme for Sulphides.— Pozzi-Escot. (*Bull. Soc. Chim.*, IX, 812) Precipitate by H_2S from dilute acid solution. Treat with HCl of 25% strength:

Solution.— Pb, Sb, Sn, Cd, Zn Add NH_4OH		*Residue.*— Cu, Mo, As, Bi, Hg, Au, Pt Treat with HNO_3	
Solution.— Cd, Zn	*Precipitate.*— Pb, Sb, Sn Dissolve in HCl	*Solution.*— Cu, Mo, As, Bi	*Residue.*— Hg, Au, Pt Dissolve in aqua regia

Nickel Dimethylglyoxime.— Ibbotson. (*Chem. News*, CIV, 224) If the precipitate is ignited quickly a considerable loss ensues, but if the paper is carefully charred by a regulated heat, which is slowly increased, the precipitate can finally be ignited to NiO and weighed as such. It is best to enclose the wet filter in two filter papers and carbonize slowly at the mouth of a muffle.

Ores — Metallurgical (Non-ferrous) Products.

Parting Platinum-Silver Buttons.— Steinmann. (*Chem. Zentr.* II, 1061) When parting the Ag button containing Pt from a Pt assay, conc. H_2SO_4 is usually employed, which dissolves some Pt. (Ordinarily 2% but in some cases as much as 5%.) By using a slightly diluted acid (5:1) and keeping the temperature at or below 250° C. this source of error is avoided.

Gold Assay — Parting.— Sterner. (*Oester, Zts.*, LIX, 461) When the Au is so small in proportion that it breaks into a powder which is collected with difficulty after dissolving off Ag, allow to settle, wash by decantation, then add a globule of Hg and boil until an amalgam is formed. After washing, dissolve off the Hg by strong HNO_3, which leaves the Au in a coherent sponge.

Cupels — Loss of Silver.— Benner and Hartmann. (*J. Ind. Eng. Chem.*, III, 805) Under the same conditions, using about 10 gm. Pb

with 40 mg. Ag, the average losses of Ag were, for the different kinds of cupels:

Kind of cupel.	Percent. Ag lost
Morganite	1.99
Casseite	3.09
Brownite	2.80
Bone ash	2.36
Cement	3.38
Equal parts cement and bone ash	2.95

So far as the experiments went, no satisfactory cupel could be made using MgO as the main constituent.

Cyanide Extraction of Gold, Control of.— White. (*J. Chem. Met. Min. Soc. S. Africa,* XII, 90) When examining the progress of extraction, by taking out samples of pulp at definite intervals, solution is going on in the samples withdrawn under more favorable conditions than in the main mass, and assays of the pulp after washing and drying do not give the information sought. Drowning out by excessive dilution is imperfect on account of adsorption or surface concentration effects. Experiments have shown that excess of $KMnO_4$, or of bleaching powder solution, immediately and completely destroy the solvent action of KCN. Formaldehyde and alkaline picrates act similarly but not immediately. The author half fills a bucket with clean water, to which he adds enough $KMnO_4$ to leave a decided color after addition of the slime sample. The sample of pulp is then taken out by use of a tube with a flap valve at the bottom, and then immediately run into the bucket under the surface of the solution with as little exposure to air as possible. Fill up with water, settle, decant, fill up again, etc., repeating this four times, in each case with addition of lime water to hasten settling. Then dry out for assay.

In discussing this communication, Mr. Crosse stated that he had known of cases in which the Au was dissolved, but was held back by the slime; in those cases the ores contained $Fe(OH)_3$.

Tin Titrations.— Morgan. (*Min. Eng. World,* Oct., 1911, 757) The reduction of Sn solutions by use of powdered Sb previous to titration is more satisfactory than in the use of Fe or Ni. This contention is illustrated by the results quoted in three separate trials by each method.

Tin in Ores.— R. G. Morgan. (*Chem. Eng.,* XIV, 289) Fuse 6 to 8 gm. NaOH in an Fe crucible. Put on top of the melt 5 gm. of the finely pulverized ore. Cover and heat slowly to fusion. Keep at dull red until the ore is decomposed. Cool, leach out with water. Keep the volume of the solution as small as possible. Rinse out the crucible with a little dilute HCl. Unite the solutions and add gradually conc. HCl until acid. Boil, settle and decant. If any undecomposed ore is visible, filter off and fuse again, or make a fresh start. To the solution add 2 or 3 horseshoe nails. Boil 15 min., and filter through glass wool into a flask. Add about 5 gm. metallic Sb, boil 5 min., then drop in a fragment of marble, and cool in a current of CO_2. Titrate with standard I solution, using starch indicator.

Volumetric for Mercury.— Smith. (*Am. Jour. Pharm.*) Methods applicable chiefly to salts used in pharmacy.

Hempel Method for Calomel.— Put 1 gm. of the substance in a stoppered 300-c.c. Erlenmeyer flask. Add 50 c.c. N/10 I solution. Move about until the material is thoroughly moistened, then add 2 gm. KI in 10 c.c. H_2O, and shake until dissolved. Titrate with N/10 $Na_2S_2O_3$; 1 c.c. N/10 = 0.02355 HgCl.

Rupp Method, Applicable to $HgCl_2$.— Dissolve 0.5 gm. in a 300-c.c. stoppered Erlenmeyer, using 10 c.c. H_2O containing 2 gm. KI. Add 25 c.c. N/ NaOH, and 6 c.c. of 40% formaldehyde. Swirl about occasionally for 10 min., and then acidulate with about 5 c.c. of 36% acetic. Add 50 c.c. N/10 I and shake well until the precipitated metallic Hg has dissolved. Titrate excess of I with N/10 $Na_2S_2O_3$; 1 c.c. N/10 I = 0.01355 gm. $HgCl_2$. Hg can be titrated by NH_4CNS similarly to the Volhard Ag titration. As this is inapplicable in presence of chlorides, its use is much limited in pharmaceutical work.

Sulfur in Roasted Blende.— Nitchie. (*J. Ind. Eng. Chem.*, IV, 30) For purposes of control in roasting blendes, it is necessary to determine the completeness of the roasting for each charge. A description of the method with details is given. The process consists in heating a fair average sample in a boat inside of a combustion tube, a current of air being passed through; the S oxides are caught and neutralized in a known amount of standard alkali, which is subsequently titrated back.

Copper by Iodide Method.— Kendall. (*J. Amer. Chem. Soc.*, XXXIII, 1947) To dissolve the Cu compound (sulfide, etc.) HNO_3 is used, affording some of the lower N oxides in the solution, which must be removed before the test can be applied. NaClO accomplishes this, but the interaction affords Cl which is also an interferer. This can be nullified by phenol, affording chlorophenol which does not interfere.

The process is conducted thus: The solution in a 300-c.c. Erlenmeyer should contain 0.1 to 0.3 gm. Cu, should have a bulk of 50 to 60 c.c. and an acidity equivalent to 4 or 5 c.c. conc. HNO_3. Its temperature should not be over 25° C. Add up to 5 c.c. of the NaClO solution and swirl to mix; when the color changes from blue to greenish, enough has been added. Let stand 2 min., then add quickly 10 c.c. phenol solution. Blow out any Cl in the atmosphere of the flask, and rinse down the sides with H_2O. Add NaOH until it gives a slight precipitate, then acidify with acetic. Add 10 c.c. of the KI solution and then titrate. The NaClO solution is made by boiling 112 gm. bleaching powder (35% Cl) with 100 gm. dry Na_2CO_3 in 1200 c.c. water. Filter and adjust strength, determined by KI and $Na_2S_2O_3$, so that 5 c.c.=30 c.c. N/10 $Na_2S_2O_3$. The phenol solution is one of 5%; NaOH of 20%; KI of 30% strength. For close titrations, the author uses $Na_2S_2O_3$ of which 1 c.c. = 0.006 Cu and finishes with one 1/6 as strong.

Lead — Electrolytic.— Fairchild. (*J. Ind. Eng. Chem.*, III, 902) A firmly adherent, smooth coating of PbO_2 can be obtained on a smooth Pt cylinder without rotation in the course of 2 hours or less, if a current of low amperage is used (slightly increased toward the end) in a hot solution (50° to 60° C.). The volume of the solution should be

200 c.c., containing 30 c.c. strong HNO_3; the anode should have 25 sq. in. of surface, and the cathode 12 sq. in. Not much over 0.1 gm. Pb should be present. Run at 0.25 amp. for 1½ hours, then raise to 0.5 amp. for the last half hour. The voltage should also be low. If much Cu is present, the amount of HNO_3 in the electrolyte should not exceed 20 c.c. conc. A solution of $PbSO_4$ in $(NH_4)C_2H_3O_2$, to which the necessary amount of HNO_3 has been added, will give satisfactory results, without any necessity for a more elaborate preparation of the electrolyte.

Manganese — Bismuthate Method.— Demorest. (*J. Ind. Eng. Chem.*, IV, 19) When Cr is present, some of it is oxidized by the bismuthate and then if $FeSO_4$ is added, and the solution back titrated with $KMnO_4$, the results are erroneous. Direct titration with Na_3AsO_3 gives accurate results. The correct method is: Dissolve 1 gm. in 45 c.c. H_2O with 15 c.c. conc. HNO_3. Boil out nitrous fumes. Cool and add bismuthate — a little at a time — until the $HMnO_4$ or MnO_2 persists on boiling. Now add KNO_2 to dissolve the MnO_2 and boil out nitrous fumes. Cool under the tap, and then add bismuthate little by little, with stirring, until 0.5 gm. has been added. Settle for a few moments, and then filter through asbestos on glass wool, and wash well. Titrate with Na_3AsO_3 solution. A brownish tint toward the end indicates insufficient acid. The standard Na_3AsO_3 is made by dissolving 2.25 gm. As_2O_3 in a hot solution of Na_2CO_3 and diluting to 2.5 liters.

Zinc in Ores.— Voigt. (*Zts. angew. Chem.*, XXIV, 2195) Treat 1 gm. (or more) with 10 c.c. HNO_3 (Gr. 1.2) and 20 to 30 c.c. conc. HCl, in an evaporating dish. Boil well, rinse into a 200-c.c. flask, add 50 to 60 c.c. NH_4OH, and boil again. When cold, fill to the mark with 1:3 NH_4OH, shake and filter. To 100 c.c. of the filtrate add a little Br to precipitate any Mn, and boil the (turbid) solution with about 1 gm. Al chips to precipitate Cu. Add 5 c.c. of Na_2SO_3 solution (1 in 4) and a little dilute NH_4OH; bring to a boil, and filter immediately. Wash the residue with hot, dilute, ammoniacal Na_2SO_3. Neutralize the filtrate with HCl, and add 10 c.c. excess of HCl (1:3). Dilute to 150 c.c., heat to boiling, and titrate hot with ferrocyanide (21.63 gm. K_4FeCy_6 and 14 gm. crystalline Na_2SO_3 per litre) using $(NH_4)_2MoO_4$ solution (9 gm. per litre) in spot tests. The end reaction is a reddish brown coloration of the spot.

Nickel Dimethylglyoxime.— Parr and Lindgren. (*Trans. Am. Brass Founders' Assoc.*, V, 120) If the solution contains sufficient tartaric to prevent the precipitation of the other metals by NH_4OH, Ni can be perfectly separated from Fe, Cr or Al. The dimethylglyoxime compound of Ni can be titrated by dissolving in standard H_2SO_4 and titrating back with KOH of corresponding strength, the end point being a yellowish tinge showing in the previously colorless solution. The values of the standard acid and alkali must be determined in terms of Ni by standardizing with pure Ni, converted to the dimethylglyoxime compound in the ordinary manner.

Nickel Separation.— Low. (*Analyst*, XXXVI, 539) Boiling an ammoniacal solution of Ni results eventually in the separation of most of the Ni as a green crystalline precipitate. In the filtrate from this, hy-

drazine hydrate will precipitate the remaining Ni by boiling. Alone, the hydrazine gives an incomplete precipitation. The indications are that the hydrazine will effect complete precipitation in presence of NH_4 salts, but NH_4OH will not completely precipitate in presence of hydrazine compounds. The precipitates obtained as above are free from alkali and can be reduced to spongy Ni by heating in a current of hydrogen.

Copper, Nickel and Cobalt.— Pedersen. (*Metallurgie,* VIII, 335) Of pyrrhotite, take 2 to 5 gm.; of mattes, 0.5 to 2.5 gm.; of slags, 10 to 20 gm. Decompose with HNO_3, then add excess of H_2SO_4 and evaporate to fumes. Add H_2O, and evaporate again. Filter off and precipitate Fe three times with NH_4OH. Dissolve, reduce and titrate. Acidify the filtrate with H_2SO_4 and separate Cu by electrolysis. Neutralize the solution with NH_4OH,, add 20 c.c. in excess, and separate Co and Ni electrolytically. Dissolve these metals from the electrode by conc. HNO_3, and evaporate off all free acid. Add a little HCl, and 150 c.c. H_2O, and precipitate Ni by dimethylglyoxime and NH_4OH. Filter, wash six times with hot H_2O, dry and weigh.

Wulfenite.— Davis. (*Met. and Chem. Eng.,* IX, 458) Decompose 1 gm. with 10 c.c. conc. HCl and 5 c.c. conc. HNO_3; add 10 c.c. H_2SO_4 and evaporate to fumes. Cool, dilute and filter off $PbSO_4$, SiO_2, etc. From this insoluble residue Pb may be dissolved out with acetate and determined in the usual way. The main filtrate is put into a pressure bottle, and H_2S passed for 15 min.; the bottle is then closed, and heated 20 min. in a boiling water-bath. Filter off the sulfide, and dissolve it in HNO_3. Evaporate to dryness and heat gently to expel H_2SO_4. Warm the dry residue with 5 c.c. dilute NH_4OH to dissolve out MoO_3. (If Cu is present, use KOH instead, and determine Cu in the insoluble residue.) Acidify the MoO_3 solution with acetic, and precipitate with excess of $Pb(C_2H_3O_2)_2$. Wash with hot $NH_4C_2H_3O_2$ to remove $PbSO_4$ and then with hot H_2O. Ignite gently and weigh $PbMoO_4$, which contains 26.15% Mo.

The filtrate from the sulfides, if colored greenish or bluish, contains V. Boil out H_2S, and oxidize by cautious addition of standard permanganate. The amount of permanganate used shows approximately the Fe and V present. Add excess of permanganate and boil for at least 5 min. Add enough $Fe(NH_4)_2(SO_4)_2$ to re-dissolve the precipitated MnO_2. Boil, then cool to 70° C. Add permanganate to pink color, then $Fe(NH_4)_2(SO_4)_2$ until a drop gives an immediate blue with ferricyanide; then run in permanganate until it destroys this property, allowing 30 sec. for the indication to develop. Then read the burette, and add permanganate until the pink color persists for 1 min. Fe standard of permanganate $\times$ 0.916 = V standard.

Tungsten in Wolframite in Presence of Molybdenite.— Trautmann. (*Z. angew. Chem.,* XXIV, 2142) The assertion is made that MoS_2 is invariably present in wolframite, from traces up to 5%. When both are brought into solution, the separation is tedious. For technical work it suffices to roast gently so as to oxidize the Mo, the W compound being unaffected, and then to extract with weak NH_4OH. The residue may be treated by any one of the processes at present in use for deter-

mination of the W. The author prefers to decompose by fusing with Na_2CO_3, in Ni crucible at high temperature.

Tellurium.— Rosenheim and Weinheber. (*Zts. anorg. Chem.*, LXIX, 266) Gutbier's method, with hydrazine, does not always give accurate results. By adding a slightly ammoniacal solution of hydrazine sulfate, and heating in an autoclave to about 130° under pressure, complete precipitation results. TeO_3 may be acidimetrically titrated in presence of glycerin, which forms with it a complex acting as a monobasic acid. The solution titrated should be half glycerin. Without glycerin the TeO_3 may be titrated with standard $Ba(OH)_2$ solution or standard NaOH to which $BaCl_2$ has been added. Under these conditions the TeO_3 acts as bibasic. A measured excess of the NaOH + $BaCl_2$ may be added, and, without filtering, the excess titrated back with standard oxalic. The indicator used is phenolphthalein.

IRON — STEEL — FERRO-ALLOYS.

Sulfur in Iron and Steel.— Elliot. (*Chem. News*, CIV, 298) The sample is annealed in admixture with ferrocyanide, previous to applying the usual evolution test. Mix 5 gm. of the sample with 0.25 gm. pulverized anhydrous ferrocyanide, wrap in paper, place in a porcelain crucible, and anneal at 750° to 850° C. in a closed muffle. When cool, examine the mass. The charred paper should still completely cover the drillings. If not, another sample must be taken. Transfer to the evolution flask, lining the neck for the time with a coil of paper to prevent any lodgment of particles there. When the flask is connected up, run in 50 c.c. conc. HCl, and carry the evolution process through as rapidly as possible. Pass gases through Cd solution and titrate with I in the usual manner.

Iron Precipitations.— Schirm. (*Chem. Ztg.*, XXXV, 897) In ferric solutions, hydrazine hydrate (N_2H_5OH) precipitates red $Fe(OH)_3$ which becomes black and granular on boiling. If the solution contains sulfates, no SO_3 is found in the precipitate. To economize the expensive reagent, the solution may be nearly neutralized with NH_4OH, and then a slight excess of hydrazine added. It is even possible to overneutralize with NH_4OH, and then add the hydrazine and boil, which will afford the black granular precipitate, containing no SO_3.

Separating Iron from Manganese.— Sanchez. (*Bull. Soc. Chim.*, IX, 880) Nearly neutralize with KOH or NaOH, add a slight excess of pyridine and boil for 10 min. Fe only is precipitated. If the amount of Mn is large, rendering it difficult to wash out, re-solution and re-precipitation in the same manner may be necessary. Zn, if present, is partially precipitated with the Fe. It may be separated in the usual manner, by redissolving and re-precipitating Fe with NH_4OH.

Iron from Vanadium.— Deiss and Leysaht. (*Chem. Ztg.*, XXXV, 869 and 878) In the ether separation, considerable amounts of V are dissolved if it is present in pentad form. When at a higher or lower valency the V is insoluble in the ether. Hence after making the ether separation, the ethereal solution should be shaken with HCl (Gr. 1.10) containing a little H_2O_2. The V then passes into the aqueous layer.

Nickel in Ferrovanadium.—Schilling. (*Chem. Ztg.*, XXXV, 1190) Ni has been found frequently in samples of ferrovanadium. To determine, treat 1 gm. of the pulverized sample with 30 c.c. HCl (Gr. 1.12). After this has apparently exerted all the solvent effect of which it is capable, add conc. HNO_3 drop by drop to complete oxidation; evaporate to dryness and heat to destroy nitrates. Re-dissolve in HCl, filter off SiO_2, washing with dilute HCl. To the filtrate add 20 c.c. of 1:5 tartaric, render faintly alkaline with NH_4OH and precipitate Ni by dimethylglyoxime..

Ferroboron.— Davis. (*Met. and Chem. Eng.*, IX, 459) Fuse 0.5 gm. with KNO_3 and Na_2CO_3 in Pt crucible; boil the melt with as little H_2O as will effect disintegration, adding a little Na_2O_2 to separate Mn. Filter. The undissolved portion may be tested for Mn by bismuthate. Acidify filtrate with HCl, then add excess of $CaCO_3$ and boil with reversed condenser to expel CO_2. Filter, cool to temperature of the laboratory, add phenolphthalein, and bring to faint alkalinity by running in N/10 NaOH from a burette. Then add 1 gm. mannitol, and titrate with the N/10 NaOH until a permanent pink (unaffected by addition of more mannitol) is obtained. 1 c.c. N/10 NaOH = 0.0011 gm. B.

Industrial Materials — Oils.

Ferric Oxide in Cement — Rapid Method.— Golubinzeff. (*Chem. Ztg.*, XXXV, 961) Put 2 gm. in an Erlenmeyer, and add 10 c.c. H_2O. Let stand 10 min. and then add 75 c.c. more. Then add gradually, with shaking, 100 c.c. of N/HCl. Shake constantly and warm somewhat until all but a little sand is dissolved. Reduce by SO_2 or H_2S, and expel the excess of reducer by a current of CO_2. Add 10 c.c. Reinhardt's solution ($MnSO_4$ with H_3PO_4) and titrate with permanganate. The method is claimed to be shorter and more accurate than the usual gravimetric methods.

Chloride in Commercial Bromides.— Herting. (*Zts. allgem. Oest. Apoth. Ver.*, XLIX, 373) Purify PbO_2 by warming with HNO_3, filtering, washing and drying. To 3 gm. of the sample add 6 gm. of the purified PbO_2 and 50 c.c. of 50% acetic, warming and shaking together from time to time, until evolution of Br ceases, and a syrupy condition begins to show. Cool, treat with H_2O and filter. To filtrate add about 20 c.c. conc. HNO_3 and 15 c.c. N/10 $AgNO_3$; filter, and titrate back excess of Ag salt by standard NH_4CNS.

In the case of NH_4Br, use 1.5 gm. sample, 12 gm. purified PbO_2, and 70 c.c. acetic.

Volatile Matter in Coals.— Parr. (*J. Ind. Eng. Chem.*, III, 900) To prevent mechanical loss on ignition for volatile matter, it has been proposed to add 10 to 15 drops of kerosene just before applying the flame. A number of results on different coals are given, both with and without this modification. In most cases the use of kerosene gives a slightly higher ash; in a few cases it is lower.

Mineral Oils in Fatty Oils.— Jablokoff. (*Rep. de Pharm.*, XXIII, 532) Mix one part of the oil with 4 of anilin. If the mixture is clear

at 20° C., mineral oil is absent. The mineral oil is insoluble in anilin. About 2% can be detected in this manner.

Carbon Disulfide in Oils.— Milan. (*C. Rend.*, CLIII, 1021) Saponify 25 gm. of the oil in the cold, with 10 c.c. conc. KOH solution. Then dissolve the soap in 150 c.c. warm H_2O, add a pinch of $NaHCO_3$ and acidify with HCl. If the escaping gases give tests for H_2S, CS_2 was present in the oil. Another method consists in mixing 50 gm. of the oil with 10 c.c. amyl alcohol and distilling. Collect the first 5 c.c. of distillate and mix in the proportion of four parts to one part of neutral cottonseed or kapok oil and a few centigrams of S. Seal in a tube and heat in boiling water. 0.1% CS_2 in the original oil will cause development of a red color. Kapok oil is superior to cottonseed oil for this test.

Turpentine, Rosin and Rosin Oil in.— Delfour. (*Bull. de Commerce*, XXXIX, 561) Prepare an alcohol solution of fuchsin in proportion of 1:100. Dilute 2 drops of this with 10 c.c. H_2O, add 1 c.c. of the turpentine, shake well and then add 2 drops NH_4OH. In presence of excess of rosin, a red precipitate appears. Further, add 20 to 30 drops NH_4OH and allow to stand. The upper layer should be colorless. If colored, it indicates presence of 3% or over of rosin or rosin oil. The limit set is 2.5 for first-class goods.

Detonators and Fulminate Compositions.— Utescher. (*Zts. Schiess. u. Sprengst.-wesen*, VI, 402) Nitro compounds should be extracted by a suitable solvent. If gunpowder is present, extract the S by CS_2, then treat with dilute H_2SO_4 to dissolve KNO_3, $KClO_3$, and Sb and Hg compounds. Remove these metals by H_2S and in the filtrate reduce chlorate to chloride by $FeSO_4$ and determine as AgCl. KNO_3 is obtained by difference. For the ordinary detonators, heat 1 gm. with 100 c.c. of 1% H_2SO_4. Filter, and through the hot filtrate pass H_2S to separate Sb and Hg. Evaporate the filtrate to dryness in a weighed dish, thereby obtaining K_2SO_4 to be calculated to $KClO_3$. The Sb and Hg sulfides are mingled with the powdered glass, etc., of the composition. They may be dissolved by use of 5 c.c. conc. HCl with 2 c.c. of 50% tartaric. An electrolytic method for the separation of the two metals is recommended.

ABSTRACTS—METALLOGRAPHY AND ALLOYS.

BY WILLIAM CAMPBELL.

Simplifications in the Technique of Metallography.— M. Le Gris. (*Rev. de Met.* 8. 1911. 335.) Small fragments of brittle material are embedded in gum lac and the whole is then polished. Small specimens can be treated in the same manner. Most etching solutions for iron and steel will dissolve the mounting, so an electrolytic method of etching is described. The specimen is touched by the cathode platinum wire.

Metallographic Notes.— H. LeChatelier. (*Rev. de Met.* 8. 1911. 367.) Study of specimens from the collection of material made by Moissan in the electric furnace, some from Charpy, also specimens of altered aluminum from Ducru. Titanium and carbide. Titaniferous cast iron. Carbide of aluminum. Quenched cast iron used by Moissan for production of artificial diamond. Charpy's steels recrystallized by annealing after working. Specimens of aluminum in various stages of decomposition, embedded in gum lac and etched by feeble current in sodium chloride solution.

Notes on the Annealing of Metals.—M. Matweeff. (*Rev. de Met.* 8. 1911. 708.) Used Le Gris' method, glass ball 1 mm. diameter and under 5 kg. Studied zinc, aluminum, copper, brass and aluminum bronze. Zinc is annealed at 100° C., copper at 300°, aluminum between 300 and 400°, brass at 500°, aluminum bronze between 450 and 550°.

Diffusion Alloys.—Le Gris. (*Rev. de Met.* 8. 1911. 613.) Extremely small samples, not over 1 mm. thick, so that the whole section can be seen at one time under the microscope. Remarkable photographs showing silver-antimony, tin-antimony, etc. Study of hardness using glass ball 0.74 mm. diameter and photographing impressions made on various parts of the diffusion alloy. Study of ternary alloys by diffusion, e.g.. tin and lead with 12% antimony.

Molybdenum-Nickel and Manganese-Thallium Alloys; also those of Calcium with Magnesium, Thallium, Lead, Copper and Silver.—N. Baar. (*Zeits. anorg. Chem.* 70. 352,) The compound MoNi forms by reaction of Mo and liquid 54% Mo at 1342° C. Ni and the compound MoNi form a eutectic at 50% Mo and 1305° C.

Mn and Tl form two conjugate liquids, solubility very small. Ca_3Mg_4 freezes at 715° C. and forms a eutectic with Ca at 78% Ca and 445° C. and with Mg at 18.5% Ca at 515° C. Calcium and lead form three compounds, Pb_3Ca, PbCa and $PbCa_2$. Two eutectics, $PbCa_2$ and Ca at 63% Ca and 700° C.; also PbCa and Pb_3Ca at 10% Ca and 625° C.

Copper and Calcium form Cu_4Ca at 933° C. This compound forms a

eutectic with Cu at 5% Ca and 910° C. With Ca at 58% Ca and 560° C. Change in solid state at 480° C.

Silver and calcium form four compounds, three eutectics.

Cerium-Tin Alloys.— Vogel. (*Zeits. anorg. Chem.* 72. 1911. 3-9.) Sn and $CeSn_2$ at 1135° C. form a eutectic, 98% Sn at 220° C. $CeSn_2$ and Ce_2Sn_3 at 1165° C. form a eutectic 60% Sn at 1103° C. Ce_2Sn_3 and Ce_2Sn at 1400° C. form a eutectic 49% Sn at 1130° C. The curve slopes down from Ce_2Sn to Ce at 830° C.

Alloys of Silver and Cadmium.— Petrenko and Fedorow. (*Zeits. anorg. Chem.* 70. 157.) The liquidus falls from the melting point of silver to that of cadmium. Six solid solutions shown on the curve. The compound AgCd forms at 200° C. in the solid.

Cobalt-Gold Alloys.— W. Wahl. (*Zeits. anorg. Chem.* 66. 60.) Cobalt (1493° C.) holds 3.5% Au in solid solution, whilst gold holds about 5½% Co in solid solution. Eutectic at 10% Co and 997° C.

Lithium Amalgams.— G. J. Zukowsky. (*Zeits, anorg. Chem.* 71. 1911. 403.) The maximum of the curve is the freezing point of LiHg at 600° C. $LiHg_2$ forms by reaction at 338° C. and $LiHg_3$ at 240° C. The eutectic of $LiHg_3$ and Hg is 1% Li at —42° C. Li and Li_3Hg form a eutectic at 162° C. and 97.6% Li. Li_3Hg forms by reaction at 379° C. and reaction points occur at 275 and 406° C.

Behavior of Lithium to Sodium, Potassium, Tin, Cadmium and Magnesium.— Masing and Tammann. (*Zeits. anorg. Chem.* 67. 183.) Li and Na, also Li and K, are but slightly soluble in the liquid state. Lithium and tin form three compounds, Li_4Sn, Li_8Sn_2 and Li_2Sn_5. Lithium and cadmium form LiCd and $LiCd_2$. Lithium and magnesium form solid solutions 85 and 95% Mg.

Silver-Sodium Diagram.— Quercigh. (*Zeits. anorg. Chem.* 68. 301.) Silver holds 13 at.% Na in solid solution. With higher percentages of Na, the alloys consist of this solid solution in a groundmass of practically pure sodium.

Alloys of Tellurium with Cadmium and Tin.— M. Kobayashi. (*Zeits. anorg. Chem.* 69. 1.) Tellurium and cadmium form a compound TeCd at 1042° C. TeSn freezes at 780.5° C. and forms a eutectic with Te, 86%, at 393° C.

Metallographic and Photochemical Research on the System Sulphur-Tellurium.— M. Chikashigé. (*Zeits. anorg. Chem.* 72, 109.) Sulphur and tellurium form a eutectic at 7% Te and 109° C. Each holds 2% of the other in solid solution. Effect of light on beta sulphur with 0.5 to 2% Te.

Gold-Magnesium Alloys.— Vogel. (*Zeits. anorg. Chem.* 63. 169.)

Auride of Magnesium.— Urasow. (Same. 64. 375.)

Gold Magnesium Diagram.— Urasow and Vogel. (Same. 67. 442.)

Vogel's curve shows three maxima, AuMg at 1160° C., $AuMg_2$ at 796° C., and $AuMg_3$ at 830° C. AuMg and Au form solid solutions. 3½ and 7½% Mg with a eutectic 4¾% Mg at 833° C. AuMg and $AuMg_2$ form a eutectic at about 17% Mg and 783° C., while $AuMg_2$ and $AuMg_3$ form a eutectic at about 23% Mg and 773° C. and $AuMg_3$ and Mg form a eutectic at about 60% Mg and 576° C.

Urasow's curve is similar but shows the formation of Au_2Mg_5. The latest diagram confirms this but Au_2Mg_5 breaks down into $AuMg_2$ and $AuMg_3$ at 720° C.

Phosphorus and Cobalt.— Zemczuzny and Schepelew. (*Zeits. anorg. Chem.* 64. 245.) Cobalt, 1500° C., and Co_2P, 1386° C., form a eutectic with 19.85 atomic percent. cobalt, freezing at 1022° C. A reaction occurs in the solid at 920° C., due to the formation of a second modification of cobalt phosphide.

Properties and Constitution of Copper-Arsenic Alloys.— Bengough and Hill. (*J. Inst. Metals.* III. 34.) Part one deals with the mechanical properties, effect of annealing, etc. The thermal diagram shows a maximum at 28.3% As or Cu_3As, 830° C. This forms a eutectic with copper at 21% As and 690° C. Copper holds 3% As in solid solution. With more arsenic we have Cu_5As_2 and changes in the solid.

Phosphor Bronze.— Hudson and Law. (*J. Inst. Metals.* III. 161.) The ternary diagram is worked out. Cu_4Sn and Cu_3P form a binary eutectic, 72% Cu, 25% Sn, 3% P. These with the copper-tin solid solution form a ternary eutectic 81% Cu, 14.2% Sn, 4.8% P, which melts at 620° C. Excellent photomicrographs illustrate the paper.

Heat Treatment of Brass, 70:30 Alloy.— Bengough and Hudson. (*Jl. Inst. Metals.* IV. 92.) Round ¾-in. hard-rolled brass used. Best combination of strength and ductility by annealing between 600 and 700° C. The effects of time of annealing were investigated, also the mechanical properties of wire at high temperatures. Microstructure of annealed, over-annealed and burnt brass is given.

Effect of Silver, Bismuth and Aluminum on Mechanical Properties of Tough-Pitch Copper Containing Arsenic.—F. Johnson. (*Jl. Inst. Metals.* IV. 163.) All three impurities have a very marked effect on properties of arsenical copper, especially on hot rolled. Effect of usual amount of silver is beneficial; bismuth is not serious up to 0.1% for rolled copper. For hot-rolled, the effect of 0.02% is noticeable Aluminum destroys hot-working properties.

Thermal Diagram of Tin with Sulphur, Selenium and Tellurium.— Biltz and Mecklenburg. (*Zeits. anorg. Chem.* 64. 226.) SnS, 21.6% S, freezes at 881° C. Melts separate into two layers: the upper is SnS, the lower contains 4½% S, plates of SnS in a groundmass of Sn.

SnSe, 60% Sn, freezes at 861° C. Alloys with less Se consist of SnSe in a groundmass of Sn. From 40 to 50% Se, we have SnSe in Sn_2Se_3. Beyond this we have $SnSe_2$ in a groundmass of Se.

SnTe, 52% Te, freezes at 796° C. With less Te we have SnTe in groundmass of Sn. SnTe and Te form a eutectic at 85% Te and 405° C.

Tin-Lead Alloys. Degens. (*Zeits. anorg. Chem.* 63. 207.) The eutectic contains 24.4 at.% of lead, and freezes at 181° C. Lead holds 12% Sn in solid solution, while tin holds 0.21 at.% lead. Change points in the solid tin at 161° C. and at 146° C., whose maximum occurs at 57 at.% Pb. (Sn_3Pb_4).

Constitution of the Ternary Alloys of Lead, Tin and Antimony.— R. Loebe. (*Metallurgie.* 8. 19 1. 7 and 33.) Gives a list of technical alloys. Reviews the binary systems, then types of ternary diagrams.

The PbSbSn system shows two reaction points, 245 and 191° C. at 10 Sn, 10 Sb, 80 Pb, and 61 Sn, 2½ Sb, 36½ Pb. The system is worked out by many sections, a great deal of thermal data, and many excellent photomicrographs.

Notes on Lead-Tin-Antimony Alloys.— W. Campbell and F. C. Elder. (SCH. MINES QUAR. 32. 244.) Discusses binary systems and gives list of technical alloys. The ternary diagram shows two reaction points, the first at 80 Pb, 10 Sb, 10 Sn at 245° C.; the second at 57½ Sn, 2½ Sb and 40 Pb, at 189° C.

Ternary System Iron-Copper-Nickel.— R. Vogel. (*Zeits. anorg. Chem.* 67. 1.) Copper and nickel form a single series of solid solutions, as do Ni and Fe. Copper and iron form two solid solutions, 3.5 and 97.25% Cu. Sections across the ternary diagram were made (1) Ni to 50% Cu, 50% Fe; (2) Cu to 75% Fe, 25% Ni; (3) Fe to 50% Cu, 50% Ni; (4) Fe to 75% Cu, 25% Ni.

Ternary Alloys of Magnesium, Zinc and Cadmium.— Bruni, Sandonnini and Quercigle. (*Zeits. anorg. Chem.* 68. 73.) The binary alloys of Zn and Cd show a eutectic at 73½ at.% Cd. and 262° C.

The alloys of Zn and Zn_2Mg (589° C.) show a eutectic at 7.5 at.% Mg and 363° C.

The alloys of Cd and $MgZn_2$ form a eutectic at 280° C. and 80 at.% Cd.

The ternary eutectic of $MgZn_2$-Zn-Cd occurs at 2 at.% Mg, 25 at.% Zn, and 73 at.% Cd, at 256° C.

Electric Conductivity and Hardness of Copper-Silver Alloys.—Kurnakow, Puschin and Senkowsky. (*Zeits. anorg. Chem.* 68. 123.) The conductivity shows a sharp drop at both the silver and the copper ends of the curve; the temperature coefficient curve is similar. Curves are straight between 4 and 90 at.% Ag.

Thermo-Electricity of Alloys.— E. Rudolfi. (*Zeits. anorg. Chem.* 67. 65.) The following alloys were investigated: Sn-Cd, Sn-Zn, Zn-Cd, Sn-Pb, Bi-Cd, Pb-Sb, Au-Ag, Au-Cu, also in part Cu-Ni and Pt-Pd. The curves at 100° and 150° C. against copper and against nickel are given.

Magnetic Properties of Alloys of Manganese, Aluminum and Copper.— Ross and Gray. (*Zeits. anorg. Chem.* 63. 349.). An alloy of 25% Mn, 12½% Al, 62½% Cu was very magnetic after pouring, without any ageing process necessary with quenched melts. Authors have previously shown that Heusler alloys show different properties from those of Cu-Mn, Mn-bronze, Mn-steel, etc., but resemble closely those of nearly pure Cu_3Al. Experiments on Al bronzes show that those consisting chiefly of Cu_3Al are most magnetic.

IRON AND STEEL

The Ovifak Iron: a Natural Steel.— C. Benedicks. (*Metallurgie,* 8. 1911. 65.) Nauckhoffs found 1.64% carbon in Ovifak iron. The author shows micrographs with cemetite and pearlite; also sulphide of iron (troilite). Very slow cooling through 700° C. is shown.

Color-Etching of Steels: Research on Austenite and Martensite.—

Robin and Gartner. (*Rev. de Met.* 8. 1911. 224.) Use picric acid in alcohol, and etch until surface of specimen is distinctly colored. Discusses effect on alloy steels. Austenite. Martensite.

Formation of Osmondite in Hypo-eutectic Steels.— J. Calian. (*Rev. de Met.* 9. 1912. 187.) Heyn and Bauer found osmondite in 0.95% C steel quenched and reheated to 400°. The author worked on steels with 0.42-, 0.65-, 0.80-, and 1.02-% C, and found that osmondite is formed between 350 and 500° C.

The Stable System Iron-Carbon.—Ruer and Iljin. (*Metallurgie*, 8. 1911. 97.) Starting with Swedish white iron, 3.75% C, five samples of gray cast iron were produced by slow cooling. These were annealed for six hours at the following temperatures: 1120, 1100, 1000, 900, 800, 700 and 600° C. and quenched, and the amounts of combined and graphitic carbon determined. The solubility line of graphite in iron was then plotted.

The Phosphides of Iron.— Konstantinon. (*Zeits. anorg. Chem.* 66. 209.) Diagram of Saklatwalla given, which shows eutectic between Fe_3P and Fe_2P. The author's diagram shows that Fe_3P forms at 1135° C. by reaction with the melt and Fe_2P. The alloys are liable to undercooling which may form a eutectic at 945° between Fe (1.7 P) and Fe_2P.

Iron-Chrome Alloys, with Special Reference to Their Resistance to Acid.— P. Monnartz. (*Metallurgie*, 8. 1911. 161 and 196.) A series of 21 alloys, 3.8 to 98% Cr, and free from carbon, were made. The freezing-point curve shows a minimum at 15% Cr and 1410° C. and a maximum at 66% Cr and 1635° C. Work of Tammann and Treitschke. Cr_2Fe corresponds to 65% Cr. A study of the behaviour of iron-chrome alloys in acid solutions follows.

Chrome Steels.— A. Portevin. (*Rev. de Met.* 8. 1911. 802.) Examined two steels of 0.12 C and Cr 13% and 17%. Sample heated to 1300° C. and cooled in 72 hours, gives structure of ferrite with tiny globules of cementite, or coarse pearlite, and is soft. Heated to 1100° C. and cooled in four hours, the structure is martensitic and hard

Tests of Steel by Corrosion.— F. Cloup. (*Rev. de Met.* 7. 1910. 605.) Study of corrosion figures by etching with (1) 4% picric acid, in alcohol; (2) Fremont's reagent: Iodine, 10; potassium iodide, 20: water, 100.

Note on the Resistance of Steel to Deformation, and the Variations in Function of Temperature.— F. Robin. (*Rev. de Met.* 8. 1911. 423.) Work on carbon steels, and special steels containing nickel, copper and aluminum. Steels were tested up to 1100° C. and in some cases to 1450° C.

Slags — Silicates — Etc.

Nature of Slag in Thomas Steel.— M. Matweieff. (*Rev. de. Met.* 7. 1910. 848.) Gives three new etching reagents: Solution of 2% ammonium oxalate colors cementite red. Boiling picrate of soda solution attacks phosphide of iron. Solution of 2% ammonium carbonate attacks phosphoric slag but does not attack silicates. Bibliography of ten titles.

The System Manganese Oxide-Silica.— Fr. Doerinckel. (*Metallur-*

gie, 8. 1912. 201.) Various concentrations of MnO and SiO_2 from 20 to 60 mol.% (17.8 to 56.1% by weight) were made and cooling curves recorded. Microstructure and thermal diagram given. Tephroite, $2MnO.SiO_2$, forms at 1345° by reaction of MnO with liquid. Rhodonite, $MnO.SiO_2$, and Tephroite form a eutectic at 45 mol.% SiO_2 at 1170° C.

Binary Systems: Li_2O-SiO_2, Li_2SiO_3-$ZnSiO_3$, etc.— H. S. Van Klooster. (*Zeits. anorg. Chem.* 69. 135.) The compound Li_4SiO_4 freezes at 1243° C.; Li_2SiO_3 at 1188° C.; and forms a eutectic with 59% SiO_2 at 1010° C. Li_2SiO_3 and SiO_2 form a eutectic at 88% SiO_2 and 948° C.

The binary system Li_2SiO_3-$ZnSiO_3$ probably forms a eutectic about 50% and 1000° C. $ZnSiO_3$ freezes at 1419° C.

The system $ZnSiO_3$-$CdSiO_3$ (1155° C.) forms a series of solid solutions with a minimum at 30% $ZnSiO_3$, 1052° C.

Binary Systems: Sodium-metasilicate with Lithium-, Magnesium-, Calcium-, Strontium- and Barium-metasilicate; also Lithium-metasilicate with Potassium-, Magnesium-, Calcium-, Strontium-, and Barium-metasilicate; and the Ternary System Na_2O-Al_2O_3-SiO_2.— R. C. Wallace. (*Zeits. anorg. Chem.* 63. 1.) The first seven systems are solid solutions generally showing a minimum point. Lithium silicate with calcium, magnesium, barium and strontium silicate forms eutectics, as well as solid solutions. The ternary diagram takes up the area Al_2O_3-$NaAlO_2$-Na_2SiO_3-SiO_2. Nephelin ($NaAlSiO_4$) and sillimanite (Al_2SiO_5).

The system $LiSiO_3$-$LiBO_2$ (843° C.) forms solid solutions; between 90 and 22% $LiSiO_3$ a eutectic with 28% $LiSiO_3$ and freezing point of 800° C. occurs.

Na_2SiO_3, 1056° C., and $NaBO_2$, 966° C., form a eutectic at 815° C. and 55% Na_2SiO_3.

Na_2SiO_3 and Na_2WO_4, 700° C., form two conjugate liquids.

In a previous paper (*Zeits. anorg. Chem.* 69. 122.) the following systems are given: $NaBO_2$ and KBO_2, 850° C., form a series of solid solutions with a minimum at 50% and 850° C. KPO_3 and KBO_2 form a compound $KPO_3.KBO_2$, freezing at 885° C. This forms a eutectic with KPO_3 (810° C.) at 90% of the latter at 680° C. and with KBO_2 (947° C.) at 30% and 770° C. $LiBO_2$ and $NaBO_2$ form a eutectic at 50% and 640° C..

Binary Systems, Calcium Metasilicate with Calcium Fluoride and Calcium Chloride.— Karandéeff. (*Zeits. anorg. Chem.* 68. 188.) $CaSiO_3$ (1501° C.) and CaF_2 (1378° C.) form a eutectic at 38.2% CaF and 1130° C. The $CaSiO_3$-$CaCl_2$ curve shows a slope from 1501° C. to the freezing point of calcium chloride at 772° C. The silicate holds 10 mol.% of chloride in solid solution.

Supplementry Bibliography.

For want of space, the following papers are noted by title only:

Photographic Recorder.— H. LeChatelier and W. Broniewski. (*Rev. de Met.* 9. 1912. 133.)

Pseudo-Binary Alloys.— A. Portevin. (*Rev. de Met.* 8. 1911. 7.)

Metallic Silicides — Action of Silicon Carbide on Metallic Oxides.— L. Baraduc-Muller. (*Rev. de Met.* 7. 1910. 657.)

Common Defects Occurring in Alloys.— Desch. (*Jl. Inst. Metals,* IV. 235.)

Electrical and Mechanical Properties of Noble-Metal Alloys.— W. Geibel. (*Zeits. anorg. Chem.* 69. 38. 70. 240.)

Magnetic Alloys Formed from Non-magnetic Materials.— A. D. Ross. (*Jl. Inst. Metals.* IV. 68.)

Crystallization of Alpha Iron.— M. Ziegler. (*Rev. de Met.* 8. 1911. 655.)

Artificial Production of Widmanstätten Structure in Carbon Steel.— N. Balaiew. (*Rev. de Met.* 7. 1910. 5·0.)

Synthesis of Meteoric Iron.—C. Benedicks. (*Rev. de Met.* 8. 1911. 85.)

Heat Treatment of Cemented Material.— L. Guillet. (*Rev. de Met.* 7. 1910. 501.)

Study of Gases Produced by Cementation Material.— DeNolly and Veyret. (*Rev. de Met.* 9. 1912. 53.)

Influence of Manganese on the Properties of Low-Carbon Steel.— George Lang. (*Metallurgie,* 8. 1911. 15 and 49.)

Microscopic Study of Brittleness of Steels, on Deformation.— F. Robin. (*Rev. de Met.* 8. 1911. 436.)

BOOK REVIEWS.

Contemporary Chemistry. By E. E. FOURNIER D'ALBE. New York, D. Van Nostrand Co., 1911. $1.25.

Works on physical chemistry by chemists usually bear a strong family resemblance to one another. In arrangement, in contents, and in illustrative data, they remind one continually of the treatises of Nernst and of Ostwald. Modifications from these types are made with hesitation, and are not always fortunate. The same selections from the limited data of twenty years ago reappear, and even data which have since been found to be wildly inaccurate (such as Le Chatelier's dissociation pressures of limestone) are reproduced.

It is refreshing, therefore, to encounter a book of a different class. The present work does not indeed profess by its title to deal with physical chemistry, but, being a book on chemistry written by a physicist, it could not help leaning strongly to this side. And it differs strikingly from other physical chemistries because its nature is determined, inevitably, by birthright, rather than acquired painfully by nurture in the schools of classical physical chemistry. Even the facts look different in the strange setting, and very often they *are* different from those that would be cited by a chemist. Several of the chapters are highly suggestive, and the volume contains in its 172 pages an immense wealth of sensations for readers who have confined themselves to works by chemists. The style is graphic, and the mode of treatment is entertaining, occasionally even diverting.

The author's knowledge of chemistry appears to be extensive as well as sound. On page 56, "chlorine," mentioned as a dissociation product of ammonium chloride, is undoubtedly a misprint for hydrogen chloride. On page 80, the statement that "no organic chemist, except Gomberg, will hear of anything else" than that carbon is quadrivalent, is hardly fair to Gomberg, and ignores the several series of compounds in which the carbon is certainly bivalent. These are, however, the only errors we have noted. The book is eminently stimulating, and every chemist should read it. A. S.

Higher Mathematics for Chemical Students. By J. R. PARTINGTON. New York, D. Van Nostrand Co., 1912. $2.00.

In these days, a knowledge of mathematics beyond the elements is indispensible to the student of chemistry. Yet, by proper selection, the portions of advanced mathematics which are most needed can be compressed into moderate compass.

In the work before us a very judicious selection has been made. A valuable feature of the book will be found in the systematic series of applications which follows each section. These include more or less purely mathematical problems, to be solved by the readers for practice. The greater number, however, are applications to specific chemical

problems, covering the whole range of physical chemistry. In a few places references are given to books or papers in which further details of the applications may be found. It would have been an advantage if such references had been made systematically throughout the book. The learner would thus have been able, first to get hold of the outline of the method, and then to consider the whole devlopment of the results from the experimental data. In this way the necessary interweaving of the mathematical theory, which is lacking in the paper, and the experimental results, which are lacking in the book, would have been facilitated. A. S.

Short Course in Graphic Statics. By CATHCART AND CHAFFEE. New York. D. Van Nostrand Co., 1912. vii + 183 pp. $1.50.

It is stated in the preface to this text that "The purpose of this book is to provide students of mechanical engineering with a brief course in graphic statics — sufficient for the solution of most of the problems met in practice by mechanical engineers — and is largely a summary of similar portions of the author's treatise 'The Elements of Graphic Statics.'" Accordingly, therefore, these subjects are chosen:

Chapter I. Force and Equilibrium Polygons.
Chapter II. Trusses; Stress Diagrams.
Chapter III. Stationary Loads; Shears and Moments.
Chapter IV. Live Loads; Shears and Moments.
Chapter V. Center of Gravity; Moment of Inertia.
Chapter VI. Friction.

With each chapter a number of examples are given which clearly show how the principles explained are to be applied. The book is very clearly written and as an introduction to the subject of graphic statics is excellent. As a text for mechanical engineering students, however, there is an unwise choice of subject matter. The full possibility of the use of the principles of graphic statics in determining force relations in machines is not brought out, and this is of vastly greater importance to the mechanical engineer than the determination of force relations in structures and of shear and moment diagrams in beams, to which a great part of the text is devoted. That part of the text which deals with the graphic determination of center of gravity and of moment of inertia of standard rolled shapes seems out of place, in that these are always available in the handbooks issued by the steel companies. Such sections as are applicable in the practice of the mechanical engineer are more easily worked up by analytical methods. This particular part of the text seems therefore to be of academic interest. There is need of a good text in the graphic determination of force relations in machines, but this one does not seem to satisfy the demand.

W. R.

Petrographic Methods. By DR. ERNST WEINSCHENK; translated by ROBERT WATSON CLARK. New York, McGraw-Hill Book Co. 396 pp., 371 illustrations. $3.50 net.

This work is a translation of the recent enlarged editions of the well-known textbooks by Dr. Weinschenk: "Anleitung zum Gebrauch des Polarisationmikskops" (third revised edition) and "Die Gesteinsbil-

denden Mineralien" (second revised edition). The avowed sole purpose of the book is the description of the rock-forming minerals and the methods of studying them. The principles of the polarizing microscope, of the more recent and convenient forms, and the methods of adjustment are first described. Then the phenomena observed with ordinary light, parallel polarized light, and convergent polarized light are explained, the various forms of apparatus and the different methods being described and elaborately illustrated. One very interesting new piece of apparatus is a chromoscope, devised by Dr.. Weinschenk, for showing simultaneously four orders of actual interference colors in place of the usual normal color charts.

The second portion of the book not only describes the rock-forming minerals, but devotes four chapters to the methods of preparation, separation, micro-chemical tests, and special reactions, and to a general discussion of the development of rock constituents. The rest of the book discusses the minerals themselves under the four headings of opaque minerals, isotropic minerals, uniaxial minerals and biaxial minerals and concludes with a key and 19 tables for determination. The work is well printed and will be of great use to students of mineralogy and petrography. A. J. M.

Cyanide Practice. By H. W. MACFARREN. New York, McGraw-Hill Book Company, 1912. x + 291 pp. Cloth, $3 net.

As stated in the preface, the purpose of this work is to furnish students, cyanide workers and those technically interested in the subject, a practical and technical exposition of the principles and basic practice applicable to cyanidation in general. The result shows that the author has succeeded admirably in accomplishing his purpose.

The practice of the cyanide process is described in such a clear and comprehensive manner that little is left to be desired. The author has carefully refrained from entering into the details of the practice of any one plant and has limited his work to fundamental principles, general practice and average results. The chapter on the suitability of an ore for cyanidation is especially good, and the manner in which different substances affect treatment is well explained. All of the various steps in the cyanide process are then thoroughly discussed in the ensuing chapters, but not at too great length or in too great detail. This book will be found a most valuable one not only for students but for instructors and workers in cyanide plants.

A most complete classified bibliography covering 55 pages is given at the end of the book, and this will be found of great assistance in looking up special points in regard to cyanide work. The work is well gotten up and contains a very complete index for easy reference.

A. L. W.

Dredges and Dredging. By CHARLES PRELINI, Professor of Civil Engineering, Manhattan College, New York. New York, D. Van Nostrand Co., 1911. 279 pp., 82 cuts. $3.00 net.

This volume falls rather below the standard of the author's former works on tunnelling, earth and rock excavation, and other civil engineering subjects. Fully three-quarters of the book consists of ma-

terial reprinted, with slight condensing and editing in some cases. from periodicals devoted to this and allied subjects. Even the illustrations of the book are reproduced from the engravings in the original publications, with all the resulting defects in clearness. However, the designers and manufacturers of this rapidly multiplying class of library furnishings deserve some gratitude from those who lack time or convenient opportunity to study the originals at the time they appear, which condition is particularly true of students in engineering colleges.

The treatment of the subject is complete, in that it refers to every known method for excavating any sort of material under water: even the removal of submarine rock ledges by mining and blasting is described, as well as the methods of drilling blast holes from floating scows. Rock breakers are also included. The several types of dredges are next described in separate chapters. In addition to the more common types of hydraulic, ladder, dipper, and grab dredges, the author selects descriptions of some less common kinds, notably the stirring dredges of various classes, and the pneumatic dredge. The single chapter on gold dredges might as well have been omitted, as being of small interest to civil engineers, while too superficial for mining men. The several methods for disposing of dredged materials are described in some detail. The chapter on cost of dredging is a collection of data from from only four sources, but as it covers the four common types of dredge, the data have a certain interest, although it would be possible to enlarge the scope and value of this chapter.

For engineering students, this book offers a concise introduction to the elements of the subject, but its value is practically limited to this field. E. K. J.

Building Stones and Clays, Their Origin, Characters and Examination. By EDWIN C. ECKEL, C.E. New York, John Wiley & Sons, 1912. 264 pp., 37 figures. $3.00 net.

This book, the author of which has long since established his reputation among the cement industries, will appeal about equally to the student, to the engineer, and to the consumer of stone. For the benefit of the student, the opening chapters of the book contain a brief, but none the less accurate, outline of those parts of geological science that bear on the occurrence and character of useful rocks. Similarly, the opening paragraphs of every chapter, of which each one relates to a single variety of building stone, review the present geological theories as to the origin of that kind of rock. For the student's benefit, also, are tabulated innumerable analyses of rocks; a possible criticism is that an undue amount of space is occupied with these analyses, especially in view of the author's rather caustic remarks concerning the reliability of rock analyses as turned out by the ordinary chemical laboratory. An exception to this criticism must be made in favor of certain tables, compiled from average results on large numbers of specimens, the object of which is to show the relationship between an original rock and its family of derived products.

The engineer will derive most profit from the chapters on the field examination of quarry sites and clay deposits, and also from those which

describe the methods of testing specimens in the laboratory. Here, too, unnecessary space is devoted to results on compressive strength of rocks, when, as everybody knows, and the author himself remarks, no stone at all suitable for building is ever subjected to its ultimate crushing pressure. In speaking of hardness, also, the author omits mention of the fact that excessive hardness not only adds nothing to the value of a stone as a building material, but is a positive detriment, in that it adds to the cost of quarrying and dressing. The bibliographies, with which each chapter is abundantly supplied, will be most useful to an engineer; by means of them he can quickly put himself in possession of all that has been published on the stone and clay resources of any district in which he may be interested. The statistics of production and market value, with which each chapter is concluded, will also prove helpful to an engineer.

For the architect, and the consumer of stone, the paragraphs describing the physical characters of the various kinds of rock will be most useful. This is particularly true of those relating to the decay and weathering of stones, the influences that cause disintegration, and the features that contribute to rapid discoloration or dissolution. Out of consideration for the users of building stones, the author has not insisted upon a theoretically exact classification of rocks, but has followed the existing traditions of the stone mason, who recognizes only two kinds of igneous rocks: granite and trap. In many other respects the author has favored local usage rather than scientific exactness. This gives the book a thoroughly practical bearing, in spite of the formidable tables of chemical analyses.

The author dedicates his work to Professor T. Nelson Dale, foremost of the scientific investigators of slates and granites in the United States, to whom also the writer of this review owes his first encouragement in the study of geology. E. K. J.

Binary Alloys. By DR. K. BORNEMANN, School of Technology, Aachen. Part II. Wilhelm Knapp, Halle a. S., 1912. Marks 9.60.

Part I was reviewed in these columns (Vol. XXXII, p. 413) and contained the thermal diagrams of the binary alloys of the alkali metals, those of copper, of silver and of gold. The present volume embraces the alloys of magnesium, calcium, zinc, cadmium, the amalgams, the alloys of aluminum, indium, thallium, tin and of lead. An appendix brings Volume I up to date. The book will be completed in three parts and will give us in very condensed form the thermal diagrams of all the binary alloys that have been worked out, besides which the text gives the chief references for each pair of alloys.

The information is valuable and has been collected chiefly from foreign periodicals devoted to metallographic research; hence the book will be welcome not only to those who have not ready access to foreign publications, but to those who have, because much time will be saved in looking up references in half a dozen different periodicals. There is but one criticism: the curves are printed on such large sheets, which have to be folded so many times, that they will rapidly become torn with

constant use. A better plan would have been to print each curve on a separate page. W. C.

The Steam Engine and Turbine. A Text-book for Engineering Colleges. By ROBERT C. H. HECK, Professor of Mechanical Engineering, Rutgers College. New York, D. Van Nostrand Company. 631 pp., 402 illustrations, tables and diagrams. $5.00 net.

A complete study of the steam engine and steam turbine will involve a consideration of a number of more or less independent subjects, the most important of which are: (1) that portion of thermodynamics dealing with heat cycles and the work done by the expansion of steam; (2) the design and proportioning of a machine which can utilize such heat cycles for the production of power; (3) an investigation of the results of experimental work performed with a view to determining the relation between the actual performance of engines and that calculated by hypothesis, and the effect of change of design upon such relations; (4) valve mechanisms and methods of governing; (5) a study of the internal forces with a view to reducing vibration and variation of angular velocity; (6) the proportioning of individual parts to withstand the stresses acting upon them.

In this book Professor Heck undertakes to deal comprehensively with all of the above subjects with the exception of the last, which is left to be included in the general subject of machine design. Th material of the book is drawn in part from the author's "Steam Engine and Other Steam Motors," with much new material added. The opening chapter contains a description of the principal types of steam engines and turbines and the functions of the essential working parts of each. The thermodynamics of a perfect gas is then taken up, and the principal heat cycles are considered on pressure-volume and temperature-entropy bases. Following this is an interesting chapter on the properties of steam, which includes a new steam table, differing slightly from that of Marks and Davis, but constructed on the same experimental data. The properties of superheated steam are presented graphically in a novel form, the coördinates being pressure and total heat content, with contours of temperature and degrees superheat, which places the information contained in a particularly convenient form for engineering work.

Having shown the peculiarities of the behavior of steam, and how it differs from a perfect gas, an analysis of the ideal steam-engine cycles is introduced, including the action of steam expanding in turbine nozzles. This leads to a comparison between the efficiency actually attained in different types of engines and that calculated from hypothesis. Considerable space is given to the discussion of missing water and the conditions affecting it. Hirn, Callender and Nicholsen, Dwelshauvers-Dery, and a number of others are quoted and the results of a large number of tests are listed, from which the author has deduced an empirical formula by which he claims that the actual water-rate of any engine in good condition may be predicted to within ten percent. It may be noted that throughout the discussion of missing water it is assumed that the leakage is negligible — an assumption which may be seriously questioned by many engineers. The section of the book de-

voted to the steam engine is concluded with two chapters dealing with the internal forces, balancing, turning effort, flywheel effect and the construction of valve gears and governors.

Two chapters are devoted to the steam turbine. In the first of these a study is made of the flow of steam through nozzles and the action of a steam jet on turbine vanes, with particular reference to the results of experimental work. In the second, a general method of turbine design is briefly laid out, followed by a description of the principal types and the details of the more important parts. The remaining chapter of the book is devoted to a short description of steam jet apparatus and condensers and air pumps. The illustrations and diagrams are clear and well chosen and the text is accompanied by many examples and problems.

It is rather difficult to judge this book as a text, since the author does not state exactly how he intends it to be used. He says in the preface that "it is assumed that the student approaches the subject with at least a general knowledge of the form and working of the steam plant and with a good preparation in the elements of physics and mechanics," that thermodynamics "is carried only so far as it is of immediate use and application" and that "certain omissions imply supplementation by other parts of the curriculum." Presumably this book presents the subject matter to be covered in a single course. From one point of view it appears desirable that the entire subject of the steam engine should be taught in one course, in order that the correlation between all of the phases may be clearly understood, but when examined in another light the advantage of this method is not so apparent. As an instance, the thermodynamics of the steam engine would probably be better understood if it were treated as part of a general course on engineering thermodynamics, together with refrigeration, air compression. heat transfer, etc., and several other subjects might be treated to better advantage separately. While the subject matter of the book forms excellent reading for the engineer, much of it appears to be too condensed for student use and it is probably the intention of the author that the text should be supplemented by lectures. The treatment of the steam turbine seems to be entirely inadequate considering the importance which this piece of mechanism has attained during the last few years, unless it is intended to be merely a preparation for a separate course on the subject. The book should be of considerable value to mechanical engineers and to teachers of mechanical engineering, and might be used as a text-book in engineering colleges whose courses are so arranged as properly to supplement it and prepare the student for its use.

H. L. P.

PERSONAL ITEMS

A. H. Case, E.M., 1905, General Manager of the Tennessee Copper Co., has been spending some time at San Pedro, N. M., reopening the company's San Pedro copper mine.

T. F. Donnelly, E.M., 1907, was recently engaged in professional work in the Cobalt district, Ontario.

S. F. Shaw, E.M., 1903, has been appointed superintendent of the Tiro General mine, owned by the American Smelting and Refining Co., at Charcas, San Luis Potosi, Mexico. J. I. Kane, E.M., 1910, is with him.

John Albert Hoag, C.E., 1910, having passed the required examinations, has received commission as Lieutenant in the United States Coast Artillery.

James McBarrow Webster, C.E., 1911, has been appointed to a position on the engineering staff of the American Sugar Refining Co., at Brooklyn, N. Y.

Franz A. Busse, C.E., 1903, has been appointed bridge engineer with the Louisville & Nashville Railroad, with offices at Louisville, Ky.

Prof. William Campbell represented the University of Durham Philosophical Society at the centennial exercises of the Philadelphia Academy of Natural Sciences, held in March, 1912.

Henry Krumb, E.M., 1898, was recently engaged in an examination of the Ohio Copper Company's mine, in Utah.

Mowry Bates, E.M., 1907, has resigned as manager of the Hughes Porcupine Mines, Ltd., and intends to pay a visit to Mexico.

Alexander P. Rogers, C.E., 1898, who recently returned from an extended professional trip through the jungle at the headwaters of the Amazon River, describes his experiences in an interesting article in *The World's Work* for April, 1912.

Henry W. Durham, C.E., 1895, who for some time past has been in charge of construction on the Cape Cod canal, has been appointed Chief Engineer in the Bureau of Highways of New York City. For four years Mr. Durham served as assistant engineer in charge of construction of one of the sections of the subway, and was then for three years in charge of the design and construction of municipal improvements in the city of Panama. The post to which Mr. Durham has been appointed has been vacant since General Bingham resigned to become Chief Engineer of the Department of Bridges.

Francis S. Foote, Jr., E.M., 1905, has been called to the University of California as Associate Professor of Railway Engineering and will take up his residence there at the beginning of the next academic year.

The Columbia alumni who are located at the University of California have just installed in their Faculty Club an excellent stained-glass window bearing the seal of Columbia, which they all agree is a pleasant reminder at lunch time. Among those who were instrumental in having the window installed were Charles Derleth, Jr., C. W. Becker, Wm. Popper, A. L. Kroeber, S. B. Christy, H. B. Torrey, T. C. Burnett, E. B. Durham and C. Kuschke.

Ray E. Walters, E.M., 1909, is resident manager of the Snowstorm mine and mill, at Mullan, Idaho.

George W. Roddewig, E.M., 1909, has been temporarily with the Coeur d'Alene Iron Works, but expected to take a position as engineer with the Snowstorm mine, Mullan, Idaho, in May, 1912.

Earl Greenough, E.M., 1910, is manager of the Marsh mine, at Burke, Idaho.

C. E. Dwyer, E.M., 1911, has been with the Morning mine, near Mullan, Idaho, since graduation.

E. P. McCrorken, E.M., 1910, was a member of an expedition sent out by a prominent exploration company of New York to investigate an alleged placer deposit in Colombia, on a branch of the Magdalena River about 600 miles from the mouth of the latter. While in the country he suffered a severe attack of malarial fever, from which he is now nearly recovered.

W. H. Weed, E.M., 1883, announces that the firm of Weed & Probert has been dissolved by mutual consent; Mr. Weed will continue his practice as consulting engineer, with office at 42 Broadway, New York.

George M. Richards, E.M., 1909, writing from the Royal Colonial Institute, Northumberland Ave., London, W.C., states that he was to leave London on March 26 for Kasindi, Belgian Congo, where he expects to remain for two years. He is to act as chief mining engineer for the Compagnie des Chemins de Fer du Congo Supérieur aux Grands Lacs Africains, and will be engaged in exploration in the district north of Lake Tanganyika and west to Lakes Kivu and Albert Edward. Richards has previously had experience in exploratory work in Alaska and Labrador.

F. A. Ayer, E.M., 1911, has been promoted to the position of chief sampler in the mines of the Detroit Copper Co., at Morenci, Ariz. He finds that this position gives him an exceptional opportunity to learn the fine points of mining geology. T. M. Bains, 1911, is with him.

A. L. Queneau, E.M., 1901, sailed for Belgium late in April, where he expects to remain for the rest of the year.

Andrew Walz, E.M., 1905, recently returned from Mexico, reports that conditions there are still unfavorable. Many Columbia men are in El Paso, waiting for an improvement in the Mexican situation.

H. C. Wilmot, E.M., 1903, has gone to the Philippines, to be with the Colorado Mining Co., Aroroy, Island of Masbate.

Don G. Caetani, E.M., 1903, recently made an investigation of the Tomboy mine and mill, at Telluride, Colo.

A. W. Evans, E.M., 1906, up to May, 1911, was assistant resident manager of the Simmer Deep mine, Johannesburg, So. Africa. He is now General Manager of the Progress and allied mines at Reefton, New Zealand.

H. N. Johnson, E.M., 1906, was married in April, 1912, at Los Angeles, Cal., and visited New York a little later. He is assistant geologist and superintendent at the Progress mine at Reefton, New Zealand, a flourishing gold quartz mine.

W. B. Tonking, E.M., 1911, is working in the Yonkers shaft for the New York City Board of Water Supply.

Charles S. Thomas, Jr., E.M., 1906, was married on April 17 in St. Louis. Thomas was in New York during the Fall and Winter as expert adviser to the District Attorney in connection with the Scheftels litigation.

J. E. Hyde, for two years Assistant in Paleontology and a former fellow in Geology, has been Assistant Professor of Stratigraphic Geology in Queens University, Kingston, Ontario, the past year, and has recently been visiting his old friends at Columbia University, accompanied by Mrs. Hyde.

Morrison B. Yung, E.M., 1901, after several years in China in charge of mining enterprises in the interior, has returned to America because of the unsettled conditions in China. Mr. Yung is accompanied by Mrs. Yung. His address for the time being is the Yale Club, New York City.

Matthew van Siclen, E.M., 1906, has recently returned from the Pedrazzini silver mines, Las Chispas, Sonora, Mexico. During the unsettled conditions in Mexico the mines have closed down so that van Siclen has returned for a visit in New York. His address is No. 44 Pine St., New York City, Room 905.

W. M. Stephen, E.M., 1911, recently made an examination of the Sunset copper mine, near Index, Wash. His office is Room 726, New York Bldg., Seattle, Wash.

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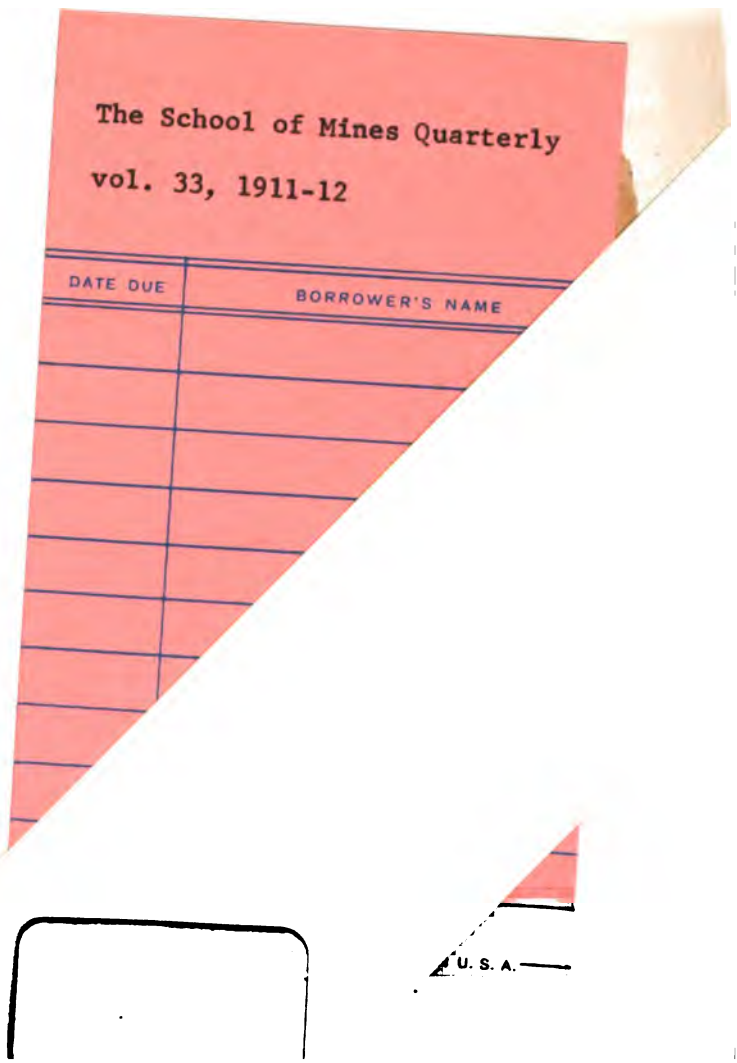
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